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DEPARTMENT OF THE INTERIOR

UNITED STATES GEOLOGICAL SURVEY

CHARLES D. WALCOTT, DIRECTOR

# PRELIMINARY REPORT

ON THE OPERATIONS OF THE

# COAL-TESTING PLANT

OF THE

UNITED STATES GEOLOGICAL SURVEY

AT THE

LOUISIANA PURCHASE EXPOSITION, ST. LOUIS, MO., 1904

EDWARD W. PARKER, JOSEPH A. HOLMES, MARIUS R. CAMPBELL
COMMITTEE IN CHARGE

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# LETTER OF TRANSMITTAL.

DEPARTMENT OF THE INTERIOR,
UNITED STATES GEOLOGICAL SURVEY,
COAL-TESTING PLANT, WORLD'S FAIR GROUNDS,
*St. Louis, Mo., January 31, 1905.*

SIR: The undersigned, appointed by you as a committee to conduct a series of tests at the United States Geological Survey coal-testing plant on the World's Fair Grounds, St. Louis, Mo., have the honor to submit the following preliminary report on the operations of the plant. The brief statements give the results obtained from September 1, 1904, when the plant was put in operation, until December 22, 1904.

The detailed report, which will be fully illustrated and contain much additional information and a discussion of the results, is in preparation and will be submitted to you at the earliest possible date.

Very respectfully,

E. W. PARKER,
J. A. HOLMES,
M. R. CAMPBELL,
*Committee.*

Hon. CHARLES D. WALCOTT,
*Director United States Geological Survey.*

# PRELIMINARY REPORT ON THE OPERATIONS OF THE COAL-TESTING PLANT OF THE UNITED STATES GEOLOGICAL SURVEY AT THE LOUISIANA PURCHASE EXPOSITION, ST. LOUIS, MO., 1904.

---

EDWARD W. PARKER,
JOSEPH A. HOLMES;
MARIUS R. CAMPBELL,
*Committee in Charge.*

---

## INTRODUCTION.

### ORGANIZATION.

The authority for conducting at the Louisiana Purchase Exposition an investigation of the coals and lignites of the United States is contained in the act of Congress providing for the urgent deficiencies in the appropriations for the fiscal year 1905, and approved February 18, 1904, as follows:

> For analyzing and testing at the Louisiana Purchase Exposition the coals and lignites of the United States, in order to determine their fuel values and the most economic method for their utilization for different purposes, under the supervision of the Director of the United States Geological Survey, thirty thousand dollars, to be available until expended: *Provided*, That all testing machinery and all coals and lignites to be tested shall be contributed without charge to the Government.

In the general deficiency bill approved April 27, 1904, an additional appropriation of $30,000 was provided, making the total sum appropriated for this work $60,000.

For carrying out the provisions of these acts the Director of the Geological Survey appointed a committee, consisting of Edward W. Parker, chairman, Joseph A. Holmes, and Marius R. Campbell, to direct the construction and operation of the plant.

It will be observed that under the law authorizing this work all of the testing machinery and all of the coals to be tested had to be furnished to the Government free of charge. Under these conditions it was not possible to equip an ideal testing plant, and the assembling and construction of such equipment as could be obtained under these provisions entailed some annoying delays in the completion of the plant. In addition to this, the delivery of a considerable quantity of operating and conveying apparatus purchased from the Link Belt Machinery Company, of Chicago, was delayed by a strike in the works of that company. As a result of these combined influences the plant was not put in operation until the first of September.

Notwithstanding these delays, the committee feels that through the hearty and patriotic cooperation of a large number of manufacturers of apparatus and machinery, it was able to collect and install, within a notably short time, a testing plant that was well suited for such pioneer work.

## BUILDINGS.

The buildings for the housing and protection of the machinery and of the operators in charge, with the exception of the chemical laboratory, were built by contract and paid for out of the appropriation. These buildings were:

(1) A boiler and engine house, in which were installed 2 Heine safety boilers, 1 Frost boiler, 1 Allis Corliss steam engine, 1 Westinghouse gas engine, 1 Bullock generator, 1 Westinghouse generator, an electric switchboard, a gas-analyzing laboratory, and all the incidental apparatus pertaining to the operations of these portions of the plant. The contract price for the construction of this building was $2,700. It was built by the Barwick Construction Company, of St. Louis.

(2) A storage and washery building, in which were installed 5 storage bins of 35 tons capacity each, 2 coal washers, 1 revolving screen, 1 set of crushing rolls, 1 pulverizer, 1 Frost steam engine, 1 hopper scale, and the conveying machinery adapted for such a plant. This building, which had to be of extra strong material and construction, was completed at a contract cost of $4,550, the Settle-Price Construction Company, of St. Louis, being the contractor.

(3) Two buildings for briquetting machines and equipment, one built by the Baerveldt Construction Company, of St. Louis, for $1,050, and the other by the Barwick Construction Company, of St. Louis, for $693.

In the letting of all contracts, bids were obtained from at least five responsible bidders.

The chemical laboratory was installed in the Metal Pavilion, a short distance from the other buildings.

## EQUIPMENT.

The plant, when put into operation in September, contained the following testing equipment, all of which had been contributed free of charge:

Two Heine water-tube boilers, 210 horsepower each, contributed by the Heine Safety Boiler Company, of New York and St. Louis.

One Allis Corliss engine, contributed by the Allis-Chalmers Company, of New York and Chicago. The Corliss engine was belt-connected to a Bullock electric generator contributed by the Bullock Electric Manufacturing Company, of Cincinnati, Ohio.

One Frost fire-tube boiler of 100 horsepower and a 50-horsepower slide-valve engine, both contributed by the Frost Engine Company, of Galesburg, Ill.

One Taylor gas producer, with complete outfit for scrubbing and purifying gas, contributed by R. D. Wood & Co., of Philadelphia.

One Westinghouse gas engine, of 235 brake horsepower, furnished by the Westinghouse Machine Company, of Pittsburg, and operated by the gas made in the Taylor producer.

One Westinghouse electric generator, belt-connected to the gas engine, contributed by the Westinghouse Electric and Manufacturing Company, of Pittsburg.

Seven motors of different horsepower, required for the operation of the different portions of the plant, contributed by the Westinghouse Electric and Manufacturing Company.

One Bartlett direct-heat rotating drier for driving out the moisture from washed coals and lignites whose moisture contents were too high to permit successful briquetting, contributed by the C. O. Bartlett & Snow Company, of Cleveland, Ohio.

Two briquetting plants—one exhibited by William Johnson & Sons, of Leeds, England, and one by the American Compressed Fuel Company, of Chicago. In order that the briquetting machine of British manufacture should be installed in conformity with the law, the freight bill was paid by the Western Coal and Mining Company, of St. Louis.

A washing plant equipped with a New Century jig, contributed by the American Concentrator Company, of Joplin, Mo., and a modified Stewart jig, contributed by the Link Belt Machinery Company, of Chicago.

The fire brick and common brick used in the construction of the coke ovens were donated by the Laclede Fire Brick Manufacturing Company, of St. Louis.

The foregoing statement includes all that might properly be considered testing machinery, but the following additional incidental equipment was contributed by the firms mentioned:

One Williams crusher, by the Williams Patent Crusher and Pulverizer Company, of St. Louis.

A complete outfit of belt-conveying apparatus, contributed and installed by the Robins Conveying Belt Company, of New York.

Scales, by Fairbanks, Morse & Co., of New York and Chicago.

Smoke-preventing device, automatic steam jet, by H. H. Hughes, of St. Louis.

Thermometers and pressure gages, by the Hohmann & Maurer Manufacturing Company, of Rochester, N. Y.

Draft gages, by the Appliance Manufacturing Company, of Chicago.

Feed-water heater and pump, by the Stilwell-Bierce Smith-Vaile Company, of Cincinnati, Ohio.

Le Chatelier pyrometers, by Charles Engelhard, of New York.

Engine indicators and attachments, by the Ashcroft Manufacturing Company, of New York, and the Crosby Steam Gage and Valve Company, of Chicago.

Metallic piston packing, by the Steel Mill Packing Company, of Detroit, Mich.

Burroughs adding machine, by the American Arithmometer Company, of St. Louis.

Insulated electric wire, by the American Steel and Wire Company, of Chicago.

Machine and engine oil for lubricating the machinery, and heavy and medium oil for binding material in the manufacture of briquettes from bitiuminous coals and lignites, by the Waters-Pierce Oil Company, of St. Louis.

Oil filter, by the Famous Filter Company, of St. Louis.

Gas meters, by the Pittsburg Meter Company, of Pittsburg.

Dumping car and bucket conveyors, by the Austin Manufacturing Company, of Chicago.

Centrifugal pump, by Henion & Hubbell, of Chicago.

Coal-tar pitch, from the Barrett Manufacturing Company, of New York and elsewhere, and the Chatfield Manufacturing Company, of Cincinnati.

Power crusher and automatic sampler for preparing samples for the chemical laboratory, by the F. W. Braun Company, of Los Angeles, Cal.

Cooking range, by the Buck Stove and Range Company, and a heating stove, by the Bridge & Beach Manufacturing Company, both of St. Louis, for the purpose of testing briquettes for domestic use.

Calculating machine, by the Keuffel & Esser Company, of New York.

The plans for the buildings and their arrangement were prepared by the Roberts & Shaefer Company, engineers, of Chicago. The construction work was done under the supervision of Mr. Bert. Fankboner, one of the engineers for the Roberts & Shaefer Company. Mr. Fankboner was accidentally killed by an electric locomotive early in October, just as the plant had been placed in successful operation and when he could have enjoyed the results of his hard and conscientious labor.

The boiler room was thoroughly equipped with standardized apparatus for weighing coal and water, for determining the force of the draft, for ascertaining the temperatures in the different parts of the furnace, and for analyzing the flue gases. The gas producer and gas engine were also provided with apparatus for ascertaining temperatures, for collecting samples of the gas as produced, and for determining the horsepower developed in the gas engine. The electric horsepower developed by both the steam engine and the gas engine was ascertained by twenty-minute readings of voltmeters and ammeters, and the electrical horsepower developed per pound of fuel consumed by the two different methods was thus accurately determined and recorded.

The chemical laboratory was equipped with every necessary apparatus and chemical material for making proximate and ultimate analyses and for determining the calorific value of the coal. The last was determined by a Mahler bomb calorimeter, and the observations were made in a room especially set aside for this purpose.

The railroad companies entering St. Louis or having coal resources along their lines have cooperated most heartily with the committee in charge of the work.

## PERSONNEL.

The various portions of the plant were put under the direction of men thoroughly equipped in their special lines. The steam test division was under the direction of Prof. L. P. Breckenridge, of the University of Illinois, with Prof. D. T. Randall, of the same university, as chief assistant. The gas-producer tests were under the direction of Prof. Robert H. Fernald, of Washington University, St. Louis, assisted by Capt. John A. Laird, also of St. Louis. The operation of the gas producer was directed by Messrs. Charles W. Lummis and C. O. Nordenson, expert engineers, of R. D. Wood & Co., of Philadelphia. The gas-engine operations were looked after by Mr. J. G. Culbertson, an engineer of the Westinghouse Company. The washing operations were under the superintendence of Mr. John D. Wick, of Chicago, general foreman of the plant. The coking plant was in charge of Mr. Fred. W. Stammler, of Johnstown, Pa., assisted by Mr. B. B. Boyd, of Uniontown, Pa. The briquetting tests were directed by Dr. Joseph Hyde Pratt, of the University of North Carolina and the North Carolina Geological Survey, assisted by Mr. A. A. Steel. The plans for the chemical laboratory, its equipment and operations, were under the direction of Prof. N. W. Lord, of the Ohio State University. His chief assistant at the coal-testing plant was Mr. E. E. Somermeier, who is also his chief assistant in his university work. The engine room was in charge of Mr. Otto Kinner, a mechanical engineer, of St. Louis, and the electrical apparatus was looked after by Mr. Jos. Underwood, also of St. Louis. Mr. Louis H. Legler, of Indiana, rendered valuable service as timekeeper and general assistant foreman of the plant.

## FIELD WORK.

The field work was placed in charge of Mr. M. R. Campbell, one of the committee appointed by the Director of the Survey for the prosecution of these tests, who was assisted throughout the field season by J. Shober Burrows and John W. Groves, and for a short time by Frank W. De Wolf. All of the coal shipped to the plant for testing, with the exception of a few carloads, was collected under the supervision of one of these Survey officials. This was done to avoid any possibility of the criticism that selected coals had been subjected to these tests.

## SAMPLING.

When the coal was shipped the Survey representative sampled it at two working faces of the mine, and the samples so obtained were shipped to the chemical laboratory for analysis. When the car was unloaded at the testing plant, what is known as a "laboratory car sample," consisting of about 200 pounds of coal, was taken from every car and sent to the laboratory, where the coal was pulverized, quartered down, and analyzed. Again, as the coal was distributed to different portions of the plant for testing, other samples were taken in a regular and uniform manner and sent to the laboratory for analysis. Such a series of coal analyses has never before been made in this country.

## REPORTS.

The preliminary reports of the operations of the different portions of the plant have been prepared by the experts in charge. A more detailed report, which will be fully illustrated by photographs, graphic charts, etc., and which will give the methods employed and results obtained in much greater detail, is in preparation, and will be published at the earliest possible date.

## RESULTS.

It will be necessary to test a larger number and greater variety of coals and lignites, and to repeat with greater care some of the tests already made, before a full discussion of the results of these investigations can be properly entered upon or their value fully appreciated. But the results already obtained from the preliminary investigations made during the last six months are sufficient to indicate their far-reaching importance in the solution of the fuel and power problems upon which the varied industries of the country depend.

Among the results already clearly indicated by these preliminary tests the following may be stated as worthy of special consideration:

(1) The tests in the steam-boiler plant of 65 carload samples of coal *from 17 States* indicate the high steam-producing capacity of American

coals, and that the quality of many of these coals may be improved by washing.

(2) Most of the American bituminous coals and lignites can be used as a source of power in a gas-producer plant.

(3) As indicated by comparative tests of 14 bituminous coals from 9 States, the power efficiency of these coals when used in the gas-producer plant is two and one-half times greater than their efficiency when used in the steam-boiler plant; or, in other words, 1 ton of these coals used in the gas-producer plant has developed, on a commercial scale, as much power as 2½ tons of the same coal when used in the ordinary steam-boiler plant.

(4) Some of the lignites from undeveloped but extensive deposits in North Dakota and Texas, when tested in the gas producer and gas engine, have shown unexpectedly high power-producing qualities, such as promise large future developments in those and other States.

(5) Some of the American coals, and the "slack" produced in mining these coals, can be briquetted on a commercial basis.

The value of the results of these investigations is, of course, not limited to the coal-producing sections of the country, but extends through every State and Territory where coal or other mineral fuel is used as a source of power. Thus, in the New England States no coal is mined; but in the year 1900 the steam power produced through the consumption of coal and used for manufacturing purposes in these States cost approximately $50,000,000. The development of this power through the more efficient methods suggested by these investigations would mean a saving to the manufacturers in these States of $15,000,000 to $20,000,000 per annum.

As another illustration of the way these investigations may influence the affairs of the nation as a whole, it may be stated that there were used in the naval vessels of the United States in 1903 approximately 500,000 tons of coal, costing $2,500,000. If the future gas producer and gas engine can be substituted on our battle ships and cruisers for the existing steam boilers and engines, the saving in the cost of coal would be not less than $1,000,000 per annum, or, what would be of far greater importance, the distances traversed by each ship without coaling would be more than doubled.

The fact that the coals and other mineral fuels used in the United States during 1904 cost the consumers approximately $1,500,000,000 indicates the magnitude and importance of the problems under investigation by the Geological Survey in connection with its coal-testing plant.

# FIELD WORK.

By Marius R. Campbell.

## INTRODUCTION.

The writer was assisted in the collection of coal for testing purposes throughout the field season by Mr. J. Shober Burrows and Mr. John W. Groves, and for a short time by Mr. Frank W. De Wolf.

In a circular letter sent out early in the season the coal operators of the United States were invited to participate in these tests, upon the conditions stated in the act of Congress under which the work was carried on, namely, that all material to be tested should be delivered at the testing plant free of cost to the Government. It was decided by the committee in charge that for practical tests of the character contemplated not less than one carload of coal should be accepted. In response to this invitation offers of coal were made from almost every coal-producing State in the Union. From these offers it was decided to accept such samples as best represented the State from which they were obtained, and also the various fields or districts within the State. Owing, however, to the fact that the testing plant was put in operation late in the season, it was found to be impossible to test all of the coal samples offered, and so a number of important coal fields and even States of large production are not represented.

It is a generally recognized principle that the value of tests upon fuel and similar materials depends largely upon the way in which the sample was obtained and what it represents. In the present work the committee in charge decided to give this feature special attention and in every case possible to superintend the collection of the sample, so as to be in a position to guarantee that the coal tested fairly represented the coal produced from the mine as far as it is possible for one carload to represent the output of a mine.

## CAR SAMPLES.

In actual operation the sampling was done as follows:

After arrangements had been made for the coal, and its transportation was assured, the representative of the testing plant visited the mine for the purpose of procuring a sample. He first entered the mine and carefully studied the character of the coal as shown in the bed before it was mined, and the method of mining and of cleaning the coal as it was loaded on cars in the mine. After becoming familiar with the underground conditions, he visited the tipple and observed the manner of handling the coal for market, getting information regarding the arrangement of the screens, the grades of coal produced, and the method of utilizing slack coal, washing, coking, etc. When this was accomplished the inspector supervised the loading of a car for testing purposes, selecting run-of-mine, lump, or nut coal, as seemed to him most nearly representative of the general output of the mine, or in particular cases selecting certain grades for special tests. The loading of the car was carefully watched so as to prevent undue picking or any irregularity that would tend to make the test sample better or worse than the average output.

In certain cases where a number of cars were in the yards, already loaded with what was regarded as representative coal, one of these cars was selected at random and shipped to the testing plant.

In case the car sample consisted of lignite or soft bituminous coal such as probably would be affected by weathering in transit, it was loaded in a box car, but the great majority of samples were shipped in ordinary open coal cars without any protection from the weather. Most of this coal was shipped during the months of August, September, October, and November, and these months were unusually dry in the vicinity of St. Louis. For this reason it seems probable that the coal in transit was subjected to little rain or snow, and that the principal change caused by weather was a change in amount of moisture due to atmospheric conditions which probably affected not only the coal in open cars but also that shipped in box cars.

### MINE SAMPLES.

As an additional check on the quality of the coal, and especially for the determination of its moisture content in its native condition, two samples were taken in each mine for chemical analysis. These were handled as follows: After a general inspection of the mine to determine the variations of the coal in thickness and quality, two points were selected in opposite or widely separated parts of the mine, and after the face of the coal was cleaned of any weathered coal or powder smoke, a cut was made across the face of the coal from roof to floor, including all of the benches of coal mined and such impurities as were not removed in ordinary work. This cut was about 3 inches wide and 1 inch deep. The coal obtained from it, amounting to 25 or 30 pounds, was caught upon an oilcloth blanket spread upon the floor of the mine so as to protect the sample from water and from including any shale or clay fragments.

The coal composing the sample was then pulverized and quartered down according to the *generally* accepted rules for preparing samples until a quart sample was obtained with the particles of coal reduced to a size not much greater than one-fourth inch in diameter. The sample was placed in an air-tight galvanized iron can having a screw top and the can was hermetically sealed by screwing the top down tight and covering the joint with tire tape. The can containing the sample was then mailed to the testing plant, and in almost all cases it reached its destination within two or three days of the date of sampling in the mine. When the can reached the chemical laboratory the sample was at once transferred to a Mason glass jar, in which it was sealed until the time for taking it up for chemical analysis.

### UNINSPECTED CAR SAMPLES.

In a few cases carload samples were shipped to the testing plant without the personal supervision of a representative of the Government. In one case this was the result of a misunderstanding; in others the sample consisted of slack coal, and its selection did not need personal supervision. In the case of the North Dakota lignite, it was impossible to send a man there early enough in the season to insure the car reaching the plant before the close of the Exposition period. This necessitated the loading of a sample without supervision, but since it is probable that the lignite from these mines is of fairly uniform quality, the sample was accepted as representative.

The committee in charge do not care to hold themselves responsible for the representative character of coal thus sent in without their personal supervision, but the results of the tests are given for what they are worth.

The samples sent in irregularly are as follows:

*Indian Territory No. 6.*—Slack coal from mine of the Southwestern Development Company, located at Coalgate, Ind. T. This sample represented the waste product of the mine, and consequently its selection did not need to be under the supervision of a representative of the testing plant.

*Kansas No. 4.*—Lump coal from mine of the Atchison Coal Mining Company, located near Atchison, Kans. Owing to misunderstanding of instructions this sample was sent without personal supervision, but its representative character is vouched for by Prof. Erasmus Haworth, State geologist of Kansas.

*Montana No. 1.*—Washed slack from Red Lodge, Mont. This is a lignitic coal which was sent in by Mr. L. S. Storrs, geologist in charge of coal lands of the Northern Pacific Railway Company.

*North Dakota No. 1.*—Brown lignite from Lehigh, N. Dak. Sent by L. S. Storrs.

*North Dakota No. 2.*—Brown lignite from mine of the Cedar Coulee Coal Company, located 4 miles southeast of Williston, N. Dak. This sample was sent to the testing plant by Mr. H. A. Storrs, engineer, Reclamation Service, Denver, Colo.

*Pennsylvania Nos. 1 and 2.*—Lump coal from Eureka No. 31 mine, of the Berwind-White Coal Mining Company, located at Windber, Somerset County, Pa. This coal was used in preliminary tests to show that the machinery was in proper working order. The tests were satisfactory, but the assumption that the coal tested was representative in character rests upon the statement of the company furnishing it.

*Pennsylvania No. 3.*—Anthracite culm from Pennsylvania Coal Company, Scranton, Pa. This sample of culm was sent for briquetting purposes and hence needed no inspection.

## DESCRIPTION OF CAR SAMPLES.

Sixty-five car samples of coal were received at the testing plant during the season, from seventeen States. These may be grouped by States as follows:

Alabama, 2; Arkansas, 6; Colorado, 1; Illinois, 6; Indiana, 2; Indian Territory, 6; Iowa, 5; Kansas, 5; Kentucky, 4; Missouri, 4; Montana, 1; New Mexico, 2; North Dakota, 2; Pennsylvania, 3; Texas, 2; West Virginia, 12; Wyoming, 2.

The following is a complete list of the car samples received, giving details of ownership, location, character of sample, etc.:

*List of car samples*

| Name of sample. | Operator. | Mine. | Location. | Railroad. |
|---|---|---|---|---|
| Alabama 1...... | Ivy Coal and Iron Co... | No. 8.......... | 1½ miles west of Horse Creek, Walker County, Ala. | Frisco System .... |
| Alabama 2...... | Galloway Coal Co., Carbon Hill, Ala. | No. 5.......... | About ¾ mile northwest of Carbon Hill, Walker County, Ala. | Frisco System .... |
| Arkansas 1...... | Central Coal and Coke Co., Kansas City, Mo. | No. 3.......... | Huntington, Sebastian County, Ark. | Frisco System .... |
| Arkansas 2...... | Central Coal and Coke Co., Kansas City, Mo. | No. 12......... | Bonanza, Sebastian County, Ark. | Frisco System .... |
| Arkansas 3...... | Western Coal and Mining Co., St. Louis, Mo. | No. 18 ........ | Jenny Lind, Sebastian County, Ark. | Missouri Pacific Railroad. |
| Arkansas 4...... | Western Coal and Mining Co., St. Louis, Mo. | No. 18 ........ | Denning, Franklin County, Ark. | Missouri Pacific Railroad. |
| Arkansas 5...... | Western Coal and Mining Co., St. Louis, Mo. | No. 4 ......... | West of Coal Hill, Franklin County, Ark. | Missouri Pacific Railroad. |
| Arkansas 6...... | Western Coal and Mining Co., St. Louis, Mo. | No. 18 ........ | Jenny Lind, Sebastian County, Ark. | Missouri Pacific Railroad. |
| Colorado 1...... | Northern Coal and Coke Co., Denver, Colo. | Simpson...... | Lafayette, Boulder County, Colo. | Colorado and Southern Railroad, Burlington System. |
| Illinois 1........ | Western Anthracite Coal and Coke Co., St. Louis, Mo. | No. 1.......... | O'Fallon, St. Clair County, Ill. | Belt Terminal R. R. of East St. Louis. |
| Illinois 2........ | Western Anthracite Coal and Coke Co., St. Louis, Mo. | No. 1.......... | O'Fallon, St. Clair County, Ill. | Belt Terminal R. R. of East St. Louis. |
| Illinois 3........ | Southern Illinois Coal Mining and Washing Co., Marion, Ill. | No. 3.......... | Marion, Williamson County, Ill. | Chicago and Eastern Illinois R. R. |
| Illinois 4........ | Donk Bros. Coal and Coke Co., St. Louis, Mo. | No. 3.......... | About 1 mile west of Troy, Madison County, Ill. | St. Louis, Troy and Eastern R. R. |
| Illinois 5........ | Donk Bros. Coal and Coke Co., St. Louis, Mo. | No. 1.......... | Collinsville, Madison County, Ill. | St. Louis, Troy and Eastern R. R. |
| Illinois 6........ | Clover Leaf Coal Co., Coffeen, Ill. | No. 1 shaft.... | Coffeen, Ill........... | Toledo, St. Louis and Western R. R. |

*received at testing plant.*

| Sampler. | Kind of coal. | Name of bed. | Tests. |
|---|---|---|---|
| M. R. Campbell ....... | Bituminous. Over 1-inch screen. | Horse Creek ...... | Steam test, p. 80.<br>Washing test, p. 66.<br>Coking test, p. 121.<br>Briquetting test, p. 148. |
| M. R. Campbell ....... | Bituminous. Through 8-inch and over ¾-inch screen, also washed slack. | Jagger............ | Steam test, p. 80.<br>Producer gas test, p. 88.<br>Coking test, p. 122. |
| M. R. Campbell ....... | Bituminous. Over 1¼-inch screen. | Huntington ...... | Steam test, p. 80.<br>Coking test, p. 122.<br>Briquetting test, p. 148. |
| M. R. Campbell ....... | Bituminous. Over 1¼-inch screen. | Jenny Lind....... | Steam test, p. 80.<br>Coking test, p. 122.<br>Briquetting test, p. 148. |
| J. W. Groves.......... | Bituminous. Over 1¼-inch screen. | Jenny Lind....... | Steam test, p. 80.<br>Coking test, p. 122.<br>Briquetting test, p. 149. |
| Shipped by operator.. | Semibituminous. Slack... | Denning.......... | Steam test, p. 80.<br>Briquetting test, p. 149. |
| J. W. Groves.......... | Semibituminous. Half lump, half slack. | Denning.......... | Steam test, p. 80.<br>Briquetting test, p. 151. |
| J. W. Groves.......... | Bituminous. Slack coal... | Jenny Lind....... | Washing test, p. 66.<br>Coking test, p. 122.<br>Briquetting test, p. 152. |
| F. W. De Wolf ........ | Black lignite. Run of mine. | ..................... | Steam test, p. 80.<br>Producer gas test, p. 90.<br>Briquetting test, p. 152. |
| M. R. Campbell ....... | Bituminous. Over 1-inch screen. | No. 8? ............ | Steam test, p. 80.<br>Coking test, p. 122.<br>Briquetting test, p. 153. |
| M. R. Campbell ....... | Bituminous. Slack........ | No. 8? ............ | Steam test, p. 80.<br>Washing test, p. 61.<br>Coking test, p. 123. |
| J. W. Groves .......... | Bituminous. Run of mine. | No. 7.............. | Steam test, p. 80.<br>Producer-gas test, p. 91.<br>Washing test, p. 66.<br>Coking test, p. 123. |
| M. R. Campbell ....... | Bituminous. Over 2-inch screen. | ..................... | Steam test, p. 80.<br>Producer-gas test, p. 93.<br>Briquetting test, p. 153. |
| M. R. Campbell ....... | Bituminous. Slack........ | ..................... | Washing test, p. 67.<br>Coking test, p. 123. |
| J. S. Burrows .......... | Bituminous. Run of mine. | No. 5.............. | Steam test, p. 80. |

*List of car samples received*

| Name of sample. | Operator. | Mine. | Location. | Railroad. |
|---|---|---|---|---|
| Indiana 1 | J. Woolley Coal Co., Evansville, Ind. | Mildred | Mildred, Sullivan County, Ind. | Evansville and Terre Haute R. R. |
| Indiana 2 | T. D. Scales Coal Co., Boonville, Ind. | Electric | Boonville, Warrick County, Ind. | Southern Rwy. |
| Indian Territory 1. | Whitehead Coal and Mining Co., Henryetta, Ind. T. | No. 1 | Henryetta, Ind. T. | Frisco System |
| Indian Territory 2. | Rock Island Coal Co., Hartshorne, Ind T. | No. 8 | Hartshorne, Ind. T. | Rock Island System. |
| Indian Territory 3. | D. Edwards & Son, Edwards, Ind. T. | No. 1 | Edwards, Ind. T. | Rock Island System. |
| Indian Territory 4. | Western Coal and Mining Co., St. Louis, Mo. | No. 5 | Half mile north of Lehigh, Ind. T. | Missouri, Kansas and Texas Rwy. |
| Indian Territory 5. | Western Coal and Mining Co., St. Louis, Mo. | No. 7 | Lehigh, Ind. T | Missouri, Kansas and Texas Rwy. |
| Indian Territory 6. | Southwestern Development Co., Parsons, Kans. | | Colgate, Ind. T. | Missouri, Kansas and Texas Rwy. |
| Iowa 1 | Anchor Coal Co., Ottumwa, Iowa. | No. 2 | Laddsdale, Iowa | Rock Island System. |
| Iowa 2 | Mammoth Vein Coal Co., Hamilton, Iowa. | No. 5 | Liberty Township, Marion County, Iowa. | Wabash Railroad. |
| Iowa 3 | Gibson Coal Mining Co., Des Moines, Iowa. | No. 4 | Near Altoona, Polk County, Iowa. | Rock Island System. |
| Iowa 4 | Centerville Block Coal Co., Centerville, Iowa. | No. 3 | Centerville, Appanoose County, Iowa. | C., B. & Q. R. R.; I. C. R. R.; C., R. I. & P. R. R.; C., M. & St. P. R. R.; |
| Iowa 5 | Inland Fuel Co., Chariton, Iowa. | No. 1 | Chariton, Lucas County, Iowa. | Burlington System, within 6 miles. |
| Kansas 1 | Western Coal and Mining Co., St. Louis, Mo. | No. 10 | Fleming, Crawford County, Kans. | Missouri Pacific R. R. |
| Kansas 2 | Western Coal and Mining Co., St. Louis, Mo. | No. 11 | Yale, Crawford County, Kans. | Missouri Pacific R. R. |
| Kansas 3 | Southern Coal and Mercantile Co., Scammon, Kans. | No. 9 | Scammon, Cherokee County, Kans. | Frisco System |

*at testing plant*—Continued.

| Sampler. | Kind of coal. | Name of bed. | Tests. |
|---|---|---|---|
| J. S. Burrows ......... | Bituminous. Run of mine. | No. 6.............. | Steam test, p. 80.<br>Producer-gas test, p. 94.<br>Washing test, p. 63.<br>Coking test, p. 123.<br>Briquetting test, p. 154. |
| J. S. Burrows ......... | Bituminous. Run of mine. | No. 5.............. | Steam test, p. 81.<br>Producer-gas test, p. 96.<br>Briquetting test, p. 154. |
| J. S. Burrows ......... | Bituminous. Over 1¾-inch screen. | Henryetta........ | Steam test, p. 81.<br>Producer-gas test, p. 98.<br>Coking test, p. 123. |
| J. W. Groves.......... | Bituminous. Run of mine. | Hartshorne....... | Steam test, p. 81.<br>Washing test, p. 67.<br>Coking test, p. 124.<br>Briquetting test, p. 155. |
| J. W. Groves.......... | Bituminous. Run of mine. | McAlester ........ | Steam test, p. 81.<br>Washing test, p. 68.<br>Coking test, p. 124.<br>Briquetting test, p. 155. |
| J. W. Groves.......... | Bituminous. Over 1-inch screen. | Lehigh ........... | Steam test, p. 81. |
| J. W. Groves.......... | Bituminous. Mixed through ¾ and ¼ inch screen. | Lehigh ........... | Washing test, p. 68.<br>Coking test, p. 124. |
| Shipped by operator.. | Bituminous. Slack........ | Lehigh ........... | Briquetting test, p. 155. |
| J. W. Groves.......... | Bituminous. Over 1¼-inch screen. | Middle bed....... | Steam test, p. 81.<br>Washing test, p. 68.<br>Coking test, p. 124. |
| J. W. Groves.......... | Bituminous. Run of mine. | Big vein.......... | Steam test, p. 81.<br>Washing test, p. 69.<br>Coking test, p. 124. |
| J. W. Groves.......... | Bituminous. Over 1½-inch screen. | Third seam....... | Steam test, p. 81.<br>Washing test, p. 69.<br>Coking test, p. 125. |
| J. W. Groves.......... | Bituminous. Over 1½-inch screen. | Lower bed........ | Steam test, p. 81.<br>Washing test, p. 69.<br>Coking test, p. 125.<br>Briquetting test, p. 158. |
| J. W. Groves ......... | Bituminous. Run of mine. | Lower bed........ | Steam test, p. 81.<br>Washing test, p. 70.<br>Coking test, p. 125. |
| J. W. Groves ......... | Bituminous. Run of mine. | Weir-Pittsburg ... | Steam test, p. 81.<br>Coking test, p. 125. |
| J. S. Burrows......... | Bituminous. Lump and nut. | Weir-Pittsburg ... | Steam test, p. 81.<br>Coking test, p. 125.<br>Briquetting test, p. 159. |
| M. R. Campbell ...... | Bituminous. Run of mine. | Weir-Pittsburg ... | Steam test, p. 81.<br>Coking test, p. 125. |

*List of car samples received*

| Name of sample. | Operator. | Mine. | Location. | Railroad. |
|---|---|---|---|---|
| Kansas 4 | The Atchison Coal Mining Co., Atchison, Kans. | Atchison | 1¼ miles below Atchison, Atchison County, Kans. | Missouri Pacific R. R. |
| Kansas 5 | Southwestern Development Co., Parsons, Kans. | No. 11 | West Mineral, Kans. | Missouri, Kansas and Texas R. R. |
| Kentucky 1 | National Coal and Iron Co., Louisville, Ky. | Straight Creek No. 2. | Straight Creek, Bell County, Ky. | Louisville and Nashville R. R. |
| Kentucky 2 | St. Bernard Mining Co., Earlington, Ky. | No. 11 | Earlington, Hopkins County, Ky. | Louisville and Nashville R. R. |
| Kentucky 3 | St. Bernard Mining Co., Earlington, Ky. | Barnsley | Barnsley, Hopkins County, Ky. | Louisville and Nashville R. R. |
| Kentucky 4 | Wheatcroft Coal and Mining Co., Wheatcroft, Ky. | Wheatcroft | Wheatcroft, Webster County, Ky. | Illinois Central R. R. |
| Missouri 1 | New Home Coal Co., Sprague, Mo. | No. 1 | New Home, Bates County, Mo. | Frisco System |
| Missouri 2 | Northwestern Coal and Mining Co., Kansas City, Mo. | No. 8 | 1 mile south of Bevier, Macon County, Mo. | Burlington System. |
| Missouri 3 | Mendota Coal and Mining Co., Mendota, Mo. | ................ | Mendota, Putnam County, Mo. | Burlington System. |
| Missouri 4 | Morgan County Coal Co., St. Louis, Mo. | ................ | Near Barnett, Morgan County, Mo. | Rock Island System. |
| Montana 1 | ................ | ................ | Red Lodge, Carbon County, Mont. | Northern Pacific R. R. |
| New Mexico 1 | American Fuel Co., Denver, Colo. | Weaver | 3 miles north of Gallup, McKinley County, N. Mex. | Santa Fe R. R. |
| New Mexico 2 | Caledonian Coal Co., Gallup, N. Mex. | Otero | 1½ miles east of Gallup, McKinley County, N. Mex. | Santa Fe R. R. |
| North Dakota 1. | ................ | ................ | Lehigh, Stark County, N. Dak. | Northern Pacific R. R. |
| North Dakota 2. | Cedar Coulee Coal Co., Williston, N. Dak. | ................ | 4 miles southeast of Williston, Williams County, N. Dak. | Great Northern R. R. |
| Pennsylvania 1. | Berwind-White Coal Mining Co., Philadelphia, Pa. | Eureka 31 | Windber, Somerset County, Pa. | Pennsylvania R. R |
| Pennsylvania 2. | Berwind-White Coal Mining Co., Philadelphia, Pa. | Eureka 31 | Windber, Somerset County, Pa. | Pennsylvania R. R |

*at testing plant*—Continued.

| Sampler. | Kind of coal. | Name of bed. | Tests. |
|---|---|---|---|
| Shipped by operator.. | Bituminous................ | ..................... | Steam test, p. 81.<br>Coking test, p. 125. |
| M. R. Campbell...... | Bituminous. Over ¼-inch screen. | Weir-Pittsburg... | Steam test, p. 81. |
| J. S. Burrows......... | Bituminous. Run of mine. | ..................... | Steam test, p. 81.<br>Coking test, p. 126.<br>Briquetting test, p. 159. |
| J. S. Burrows......... | Bituminous. Over ¼-inch screen. | No. 11............ | Steam test, p. 81.<br>Coking test, p. 126.<br>Briquetting test, p. 159. |
| J. S. Burrows......... | Bituminous. Run of mine and coal over ¼-inch screen. | No. 9............. | Steam test, p. 81.<br>Producer-gas test, p. 99.<br>Washing test, p. 70.<br>Coking test, p. 126. |
| J. S. Burrows......... | Bituminous. Run of mine. | No. 11............. | Steam test, p. 81.<br>Washing test, p. 70.<br>Coking test, p. 126. |
| J. S. Burrows......... | Bituminous. Run of mine. | ..................... | Steam test, p. 81.<br>Briquetting test, p. 160. |
| J. W. Groves.......... | Bituminous. Run of mine. | Bevier............ | Steam test, p. 81.<br>Producer gas test, p. 101.<br>Washing test, p. 70.<br>Coking test, p. 126. |
| Shipped by operator.. | Bituminous slack.......... | ..................... | Steam test, p. 82.<br>Washing test, p. 64.<br>Coking test, p. 126. |
| J. W. Groves.......... | Bituminous. Run of mine. | ..................... | Steam test, p. 82.<br>Coking test, p. 126. |
| Shipped by L. S. Storrs | Black lignite (?) No. 4, washed nut. | ..................... | Producer gas test, p. 102.<br>Briquetting test, p. 160. |
| M. R. Campbell....... | Black lignite. Run of mine | No. 3 and No. 3½.. | Steam test, p. 82.<br>Briquetting test, p. 161. |
| M. R. Campbell....... | Black lignite. Slack...... | Otero, Thatcher, Crownpoint. | Steam test, p. 82.<br>Briquetting test, p. 162. |
| Shipped by L. S. Storrs | Brown lignite ............ | ..................... | Steam test, p. 82.<br>Briquetting test, p. 162. |
| Shipped by operator.. | Brown lignite............. | ..................... | Producer-gas test, p. 104. |
| Shipped by operator.. | Bituminous................ | B (?)............. | Steam test, p. 82. |
| Shipped by operator.. | Bituminous................ | B (?)............. | Steam test, p. 82. |

*List of car sample received*

| Name of sample. | Operator. | Mine. | Location. | Railroad. |
|---|---|---|---|---|
| Pennsylvania 3. | Pennsylvania Coal Co., Scranton, Pa. | | | |
| Texas 1 | Houston County Coal and Manufacturing Co., Crockett, Tex. | Wootters | 11 miles south of Crockett, Houston County, Tex. | International and Great Northern R. R. |
| Texas 2 | Consumers' Lignite Co., Dallas, Tex. | | Hoyt, Wood County, Tex. | Texas Short Line, M., K. & T. R. R. |
| West Virginia 1. | Virginia and Pittsburg Coal and Coke Co., New York, N. Y. | Kingmont | Kingmont, Marion County, W. Va. | Baltimore and Ohio R. R. |
| West Virginia 2. | Pitcairn Coal Co., Clarksburg, W. Va. | Pitcairn | Clarksburg, Harrison County, W. Va. | Baltimore and Ohio R. R. |
| West Virginia 3. | West Virginia Coal Co., Morgantown, W. Va. | Richard | 4 miles southeast of Morgantown, Monongalia County, W. Va. | Morgantown and Kingwood R. R. |
| West Virginia 4. | West Virginia Coal Co., Morgantown, W. Va. | Bretz | 4 miles northwest of Kingwood, Preston County, W. Va. | Morgantown and Kingwood R. R. |
| West Virginia 5. | Davis Colliery Co., Elkins, W. Va. | Coalton | Coalton, Randolph County, W. Va. | Coal and Coke R. R. |
| West Virginia 6. | The New River Smokeless Coal Co., Rush Run, W. Va. | Rush Run | Rush Run, Fayette County, W. Va. | Chesapeake and Ohio R. R. |
| West Virginia 7. | The New River Smokeless Coal Co., Rush Run, W. Va. | Sun No. 1 | Sun, Fayette County, W. Va. | Chesapeake and Ohio R. R. |
| West Virginia 8. | The Gauley Mountain Coal Co., Ansted, W. Va. | Gauley Mountain. | Ansted, Fayette County, W. Va. | Chesapeake and Ohio R. R. |
| West Virginia 9. | Mount Carbon Coal Co., (Limited), Powellton, W. Va. | Vulcan | 1 mile above Powellton, Fayette County, W. Va. | Chesapeake and Ohio R. R. |
| West Virginia 10 | Stuart M. Buck, Bramwell, W. Va. | Experimental mine. | Mora, Mercer County, W. Va. | Norfolk and Western R. R. |
| West Virginia 11 | W. H. Coffman, Bluefield, W. Va. | Zenith 1 and 2. | Zenith, McDowell County, W. Va. | Norfolk and Western R. R. |
| West Virginia 12. | Big Sandy Coal and Coke Co., Marytown, W. Va. | Big Sandy | Big Sandy, McDowell County, W. Va. | Norfolk and Western Railroad. |
| Wyoming 1 | Wyoming Coal Mining Co., Monarch, Wyo. | Monarch | 9 miles northwest of Sheridan, Sheridan County, Wyo. | Burlington System. |
| Wyoming 2 | Cambria Fuel Co., Cambria, Wyo. | Antelope 1 and 2 and Jumbo. | Cambria, Weston County, Wyo. | Burlington System. |

*at testing plant*—Continued.

| Sampler. | Kind of coal. | Name of bed. | Tests. |
|---|---|---|---|
| Shipped by operator.. | Anthracite. Culm ........ | .................... | Steam test, p. 82.<br>Briquetting test, p. 163. |
| M. R. Campbell....... | Brown lignite.............. | .................... | Producer-gas test, p. 106. |
| M. R. Campbell....... | Brown lignite.............. | .................... | Producer-gas test, p. 107. |
| J. S. Burrows......... | Bituminous. Run of mine. | Pittsburg......... | Steam test, p. 82.<br>Producer-gas test, p. 109.<br>Coking test, p. 126. |
| J. S. Burrows......... | Bituminous. Run of mine. | Pittsburg......... | Steam test, p. 82.<br>Washing test, p. 71.<br>Coking test, p. 127. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Upper Freeport... | Steam test, p. 82.<br>Washing test, p. 71.<br>Coking test, p. 127.<br>Briquetting test, p. 165. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Upper Freeport... | Steam test, p. 82.<br>Producer-gas test, p. 111.<br>Washing test, p. 71.<br>Coking test, p. 127. |
| J. S. Burrows.......... | Bituminous. Over 1¼ inch screen. | Upper Freeport... | Steam test, p. 82.<br>Washing test, p. 72.<br>Coking test, p. 127. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Quinnimont...... | Steam test, p. 82.<br>Coking test, p. 128.<br>Briquetting test, p. 165. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Sewall............ | Steam test, p. 82.<br>Coking test, p. 128. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Ansted .......... | Steam test, p. 82.<br>Coking test, p. 128. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Powellton ........ | Steam test, p. 82.<br>Producer-gas test, p. 112.<br>Washing test, p. 72.<br>Coking test, p. 128. |
| J. S. Burrows.......... | Bituminous. Over ¾-inch screen. | No. 6 (?) .......... | Steam test, p. 83.<br>Coking test, p. 128. |
| J. S. Burrows.......... | Bituminous. Run of mine. | Pocahontas....... | Steam test, p. 83.<br>Coking test, p. 129. |
| J. S. Burrows.......... | Bituminous. Run of mine. | No. 8............... | Steam test, p. 83.<br>Producer-gas test, p. 114.<br>Washing test, p. 72.<br>Coking test, p. 129. |
| F. W. De Wolf ........ | Bituminous. Over 5-inch screen. | .................... | Steam test, p. 83.<br>Briquetting test, p. 165. |
| F. W. De Wolf ........ | Bituminous. Run of mine | .................... | Steam test, p. 83.<br>Producer-gas test, p. 115. |

# WORK OF THE CHEMICAL LABORATORY.

By N. W. Lord.

## INTRODUCTION.

In this preliminary report only the mine samples and the laboratory car samples of the coals are considered.

Numerous analyses made in connection with steam tests, producer-gas tests, coking tests, etc., representing the portions of coal used in these tests, are published in the several reports giving the results of the tests and are not repeated here.

## SAMPLING.

The mine samples were taken by the agents of the Survey at the mines from which the coal was shipped and were sent to the laboratory in sealed cans. They were taken on opposite sides of the mine or in places widely separated, so that to a certain extent they show the variations in the quality of the coal within the limits of the mine. In the final report the points at which the samples were obtained will be accurately described, but in the present report the samples will be designated merely "mine sample A" and "mine sample B."

The laboratory car samples were taken from the carload of coal sent to the plant for testing at the time that it was unloaded. They usually represent either the whole car or that portion of it used in the tests. The laboratory car sample was obtained after the coal had been passed through rolls having an aperture of about 1¼ inches, by taking portions at short intervals from the buckets of the conveyor. These portions, after thorough mixing, were used for the preparation of the laboratory sample.

## LOSS ON AIR DRYING.

Most coals rapidly lose moisture on exposure to air. In grinding the large sample down to the fine state of pulverization necessary in the small laboratory sample there is liable to be a large and undetermined loss of moisture. To reduce the error due to this loss of moisture, each sample of coal, after a rapid preliminary crushing to about one-fourth inch size and reduction in bulk by quartering where desirable, was weighed and then exposed to the air for about twenty-four hours, or until the loss of weight on further exposure became slight.

The loss of weight thus determined constitutes the "loss on air drying." The coal in this air-dried condition was then pulverized for the final sample to be used in the various operations of the laboratory.

The reduction of the actual results obtained upon the air-dried sample to the equivalent results on the original sample before air drying was accomplished as follows: Each of the results upon the air-dried sample was multiplied by the fraction the air-dried coal formed of the original coal and then (in the proximate analyses) the percentage loss on air drying was added to the figure for moisture so obtained.

In the case of an ultimate analysis the figures so obtained for the oxygen and the hydrogen were each increased by amounts equivalent to the oxygen and hydrogen represented by the loss on air drying when considered as water.

## METHODS OF ANALYSIS.

The methods of analysis were essentially those recommended by the committee of the American Chemical Society. The moisture was determined by drying the weighed sample for one hour in an air bath at 105° C. The calorific value was determined in the Mahler bomb calorimeter. The actual value of the result in the calorimeter was corrected for the sulphuric acid formed in the bomb.

In calculating the calorific value from the ultimate analysis the calorific values of hydrogen, carbon, and sulphur were taken as, respectively, 34,460, 8,080, and 2,250 calories.

## PERSONNEL.

Mr. E. E. Somermeier, of the department of metallurgy of the Ohio State University, was chief chemist and had immediate charge of the laboratory at the testing plant. The assistant chemists employed in the laboratory were Mr. F. M. Stanton, Mr. John H. Crawford, jr., and Mr. G. A. Burrell. Mr. F. A. Bryan was employed in collecting the car samples and washery samples at the plant.

## ANALYSES OF MINE AND LABORATORY CAR SAMPLES.

The following table gives in a condensed form the analyses of "mine" and "laboratory car" samples arranged according to States:

*Analyses of mine and laboratory car samples of coal.*

| | | Alabama No. 1. | | | Alabama No 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Labora-tory car sample. | Mine sample A. | Mine sample B. | Labora-tory car sample. |
| | Laboratory sample No | 1077 | 1078 | 1201 | 1075 | 1076 | 1225 |
| | Loss of moisture on air drying | ......... | ......... | 0.80 | ......... | ......... | 0.80 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | ......... | ......... | 1.55 | ......... | ......... | 2.58 |
| | Volatile matter | ......... | ......... | 32.10 | ......... | ......... | 33.15 |
| | Fixed carbon | ......... | ......... | 53.71 | ......... | ......... | 51.74 |
| | Ash | ......... | ......... | 12.64 | ......... | ......... | 12.53 |
| Ultimate | Sulphur | ......... | ......... | 0.73 | ......... | ......... | 1.02 |
| | Hydrogen | ......... | ......... | 4.96 | ......... | ......... | 4.79 |
| | Carbon | ......... | ......... | 72.16 | ......... | ......... | 69.24 |
| | Nitrogen | ......... | ......... | 1.66 | ......... | ......... | 1.55 |
| | Oxygen | ......... | ......... | 7.85 | ......... | ......... | 10.87 |
| | Calorific value determined: | | | | | | |
| | Calories | ......... | ......... | 7,199 | ......... | ......... | 6,916 |
| | British thermal units | ......... | ......... | 12,958 | ......... | ......... | 12,449 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ......... | ......... | 7,218 | ......... | ......... | 6,799 |
| | British thermal units | ......... | ......... | 12,992 | ......... | ......... | 12,238 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 1.22 | 1.35 | 2.34 | 2.25 | 2.42 | 3.36 |
| | Volatile matter | 31.53 | 31.67 | 31.84 | 35.70 | 34.83 | 32.86 |
| | Fixed carbon | 54.44 | 53.35 | 53.28 | 53.01 | 51.62 | 51.33 |
| | Ash | 12.81 | 13.63 | 12.54 | 9.04 | 11.13 | 12.43 |
| Ultimate | Sulphur | 0.71 | 0.71 | 0.72 | 1.09 | 1.10 | 1.01 |
| | Hydrogen | ......... | ......... | 5.01 | ......... | ......... | 4.84 |
| | Carbon | ......... | ......... | 71.58 | ......... | ......... | 68.69 |
| | Nitrogen | ......... | ......... | 1.65 | ......... | ......... | 1.54 |
| | Oxygen | ......... | ......... | 8.50 | ......... | ......... | 11.49 |
| | Calorific value determined: | | | | | | |
| | Calories | ......... | 7,217 | 7,142 | 7,296 | 7,053 | 6,861 |
| | British thermal units | ......... | 12,991 | 12,856 | 13,133 | 12,695 | 12,350 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ......... | ......... | 7,160 | ......... | ......... | 6,745 |
| | British thermal units | ......... | ......... | 12,888 | ......... | ......... | 12,141 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Arkansas No 1. | | | Arkansas No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| | Laboratory sample No | 1045 | 1046 | 1114 | 1049 | 1053 | 1160 |
| | Loss of moisture on air drying | ........ | ........ | 2.10 | ........ | ........ | 1.50 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate. | Moisture | ........ | ........ | 1.17 | ........ | ........ | 0.74 |
| Proximate. | Volatile matter | ........ | ........ | 17.83 | ........ | ........ | 16.26 |
| Proximate. | Fixed carbon | ........ | ........ | 68.12 | ........ | ........ | 73.66 |
| Proximate. Ultimate. | Ash | ........ | ........ | 12.88 | ........ | ........ | 9.34 |
| Proximate. Ultimate. | Sulphur | ........ | ........ | 1.27 | ........ | ........ | 1.90 |
| Ultimate. | Hydrogen | ........ | ........ | 4.00 | ........ | ........ | 4.13 |
| Ultimate. | Carbon | ........ | ........ | 75.68 | ........ | ........ | 80.03 |
| Ultimate. | Nitrogen | ........ | ........ | 1.47 | ........ | ........ | 1.40 |
| Ultimate. | Oxygen | ........ | ........ | 4.70 | ........ | ........ | 3.20 |
| | Calorific value determined: | | | | | | |
| | Calories | ........ | ........ | 7,450 | ........ | ........ | 7,756 |
| | British thermal units | ........ | ........ | 13,410 | ........ | ........ | 13,961 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 7,319 | ........ | ........ | 7,794 |
| | British thermal units | ........ | ........ | 13,174 | ........ | ........ | 14,029 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate. | Moisture | 1.02 | 0.75 | 3.24 | 0.95 | 0.78 | 2.23 |
| Proximate. | Volatile matter | 17.88 | 18.50 | 17.46 | 18.70 | 16.60 | 16.02 |
| Proximate. | Fixed carbon | 73.61 | 73.77 | 66.69 | 73.38 | 73.53 | 72.55 |
| Proximate. Ultimate. | Ash | 7.49 | 6.98 | 12.61 | 6.97 | 9.09 | 9.20 |
| Proximate. Ultimate. | Sulphur | 1.10 | 1.15 | 1.24 | 2.12 | 2.50 | 1.87 |
| Ultimate. | Hydrogen | ........ | ........ | 4.15 | ........ | ........ | 4.24 |
| Ultimate. | Carbon | ........ | ........ | 74.09 | ........ | ........ | 78.83 |
| Ultimate. | Nitrogen | ........ | ........ | 1.44 | ........ | ........ | 1.38 |
| Ultimate. | Oxygen | ........ | ........ | 6.47 | ........ | ........ | 4.48 |
| | Calorific value determined: | | | | | | |
| | Calories | 8,019 | ........ | 7,294 | 7,993 | ........ | 7,639 |
| | British thermal units | 14,434 | ........ | 13,129 | 14,387 | ........ | 13,750 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 7,165 | ........ | ........ | 7,677 |
| | British thermal units | ........ | ........ | 12,897 | ........ | ........ | 13,819 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Arkansas No. 3. | | | Arkansas No. 5. | | |
|---|---|---|---|---|---|---|---|
| | | **Mine sample A.** | **Mine sample B.** | **Laboratory car sample.** | **Mine sample A.** | **Mine sample B.** | **Laboratory car sample.** |
| **Laboratory sample No** | | 1115 | 1118 | 1296 | 1130 | 1131 | 1331 |
| **Loss of moisture on air drying** | | 0.80 | 0.80 | 1.40 | 0.70 | 1.30 | 1.10 |
| ***Analysis of air-dried sample.*** | | | | | | | |
| Proximate | **Moisture** | 0.81 | 0.84 | 0.80 | 0.68 | 0.51 | 1.28 |
| | **Volatile matter** | 17.54 | 16.81 | 19.75 | 14.87 | 15.19 | 12.82 |
| | **Fixed carbon** | 73.68 | 69.59 | 67.65 | 77.45 | 76.94 | 73.69 |
| | **Ash** | 7.97 | 12.76 | 11.80 | 7.00 | 7.36 | 12.21 |
| Ultimate | **Sulphur** | 1.43 | 1.47 | 1.30 | 1.58 | 1.97 | 2.01 |
| | **Hydrogen** | .......... | .......... | 4.07 | .......... | .......... | 3.74 |
| | **Carbon** | .......... | .......... | 76.37 | .......... | .......... | 77.29 |
| | **Nitrogen** | .......... | .......... | 1.55 | .......... | .......... | 1.89 |
| | **Oxygen** | .......... | .......... | 4.91 | .......... | .......... | 3.36 |
| **Calorific value determined:** | | | | | | | |
| **Calories** | | 7,931 | .......... | 7,586 | 8,017 | .......... | 7,448 |
| **British thermal units** | | 14,275 | .......... | 13,655 | 14,431 | .......... | 13,406 |
| **Calorific value calculated from ultimate analysis:** | | | | | | | |
| **Calories** | | .......... | .......... | 7,393 | .......... | .......... | 7,434 |
| **British thermal units** | | .......... | .......... | 13,307 | .......... | .......... | 13,381 |
| ***Analysis corrected to sample as received.*** | | | | | | | |
| Proximate | **Moisture** | 1.60 | 1.63 | 2.19 | 1.38 | 1.80 | 2.36 |
| | **Volatile matter** | 17.40 | 16.68 | 19.47 | 14.76 | 15.00 | 12.68 |
| | **Fixed carbon** | 73.09 | 69.03 | 66.71 | 76.91 | 75.94 | 72.88 |
| | **Ash** | 7.91 | 12.66 | 11.63 | 6.95 | 7.26 | 12.08 |
| Ultimate | **Sulphur** | 1.42 | 1.46 | 1.28 | 1.52 | 1.94 | 1.99 |
| | **Hydrogen** | .......... | .......... | 4.17 | .......... | .......... | 3.82 |
| | **Carbon** | .......... | .......... | 75.31 | .......... | .......... | 76.44 |
| | **Nitrogen** | .......... | .......... | 1.53 | .......... | .......... | 1.37 |
| | **Oxygen** | .......... | .......... | 6.08 | .......... | .......... | 4.30 |
| **Calorific value determined:** | | | | | | | |
| **Calories** | | 7,868 | .......... | 7,480 | 7,961 | .......... | 7,366 |
| **British thermal units** | | 14,162 | .......... | 13,464 | 14,330 | .......... | 13,259 |
| **Calorific value calculated from ultimate analysis:** | | | | | | | |
| **Calories** | | .......... | .......... | 7,289 | .......... | .......... | 7,353 |
| **British thermal units** | | .......... | .......... | 13,120 | .......... | .......... | 13,235 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Arkansas No. 6. | Colorado No. 1. | | |
|---|---|---|---|---|---|
| | | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| | Laboratory sample No | 1542 | 1383 | 1397 | 1523 |
| | Loss of moisture on air drying | 3.00 | 3.90 | 4.00 | 6.00 |
| | *Analysis of air-dried sample.* | | | | |
| Proximate | Moisture | 0.82 | 16.77 | 18.58 | 13.49 |
| | Volatile matter | 14.32 | 35.18 | 35.42 | 37.11 |
| | Fixed carbon | 70.62 | 44.29 | 42.38 | 43.03 |
| | Ash | 14.24 | 3.76 | 3.62 | 6.37 |
| Ultimate | Sulphur | 1.30 | 0.54 | 0.48 | 0.58 |
| | Hydrogen | | | | 5.75 |
| | Carbon | | | | 61.13 |
| | Nitrogen | | | | 1.22 |
| | Oxygen | | | | 24.95 |
| | Calorific value determined: | | | | |
| | Calories | | 5,918 | | 5,995 |
| | British thermal units | | 10,652 | | 10,791 |
| | Calorific value calculated from ultimate analysis: | | | | |
| | Calories | | | | 5,859 |
| | British thermal units | | | | 10,546 |
| | *Analysis corrected to sample as received.* | | | | |
| Proximate | Moisture | 3.80 | 20.02 | 21.84 | 18.68 |
| | Volatile matter | 13.89 | 33.81 | 34.00 | 34.88 |
| | Fixed carbon | 68.50 | 42.56 | 40.68 | 40.45 |
| | Ash | 13.81 | 3.61 | 3.48 | 5.99 |
| Ultimate | Sulphur | 1.26 | 0.52 | 0.46 | 0.55 |
| | Hydrogen | | | | 6.07 |
| | Carbon | | | | 57.46 |
| | Nitrogen | | | | 1.15 |
| | Oxygen | | | | 28.78 |
| | Calorific value determined: | | | | |
| | Calories | | 5,687 | | 5,635 |
| | British thermal units | | 10,237 | | 10,143 |
| | Calorific value calculated from ultimate analysis: | | | | |
| | Calories | | | | 5,507 |
| | British thermal units | | | | 9,913 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Illinois No. 1. | | | Illinois No. 2. | Illinois No. 3. | | |
|---|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| Laboratory sample No. | | 1095 | 1096 | 1261 | 1152 | 1170 | 1171 | 1318 |
| Loss of moisture on air-drying. | | 4.40 | 3.20 | 3.70 | 7.10 | 1.50 | 1.80 | 2.70 |
| *Analysis of air-dried sample.* | | | | | | | | |
| Proximate | Moisture | 7.08 | 7.09 | 6.28 | 5.31 | 6.00 | 5.63 | 5.96 |
| | Volatile matter | 41.12 | 41.66 | 38.92 | 34.29 | 32.16 | 34.93 | 30.29 |
| | Fixed carbon | 41.00 | 40.85 | 41.08 | 36.24 | 54.49 | 51.78 | 52.16 |
| | Ash | 10.80 | 10.40 | 13.72 | 24.16 | 7.26 | 7.67 | 11.59 |
| Ultimate | Sulphur | 4.41 | 4.17 | 4.25 | 4.30 | 1.00 | 2.08 | 1.77 |
| | Hydrogen | ......... | ......... | 5.09 | 4.57 | ......... | ......... | 4.92 |
| | Carbon | ......... | ......... | 62.01 | 54.06 | ......... | ......... | 67.30 |
| | Nitrogen | ......... | ......... | 1.07 | 0.78 | ......... | ......... | 1.43 |
| | Oxygen | ......... | ......... | 13.86 | 12.13 | ......... | ......... | 12.99 |
| Calorific value determined: | | | | | | | | |
| | Calories | 6,521 | ......... | 6,360 | 5,471 | 6,986 | ......... | 6,724 |
| | British thermal units | 11,738 | ......... | 11,448 | 9,848 | 12,565 | ......... | 12,103 |
| Calorific value calculated from ultimate analysis: | | | | | | | | |
| | Calories | ......... | ......... | 6,263 | 5,516 | ......... | ......... | 6,615 |
| | British thermal units | ......... | ......... | 11,273 | 9,929 | ......... | ......... | 11,907 |
| *Analysis corrected to sample as received.* | | | | | | | | |
| Proximate | Moisture | 11.17 | 10.06 | 9.75 | 12.03 | 7.50 | 7.34 | 8.50 |
| | Volatile matter | 39.31 | 40.33 | 37.48 | 31.86 | 31.68 | 34.29 | 29.47 |
| | Fixed carbon | 39.20 | 39.54 | 39.57 | 33.67 | 53.67 | 50.84 | 50.75 |
| | Ash | 10.32 | 10.07 | 13.20 | 22.44 | 7.15 | 7.53 | 11.28 |
| Ultimate | Sulphur | 4.22 | 4.04 | 4.10 | 4.00 | 0.99 | 2.04 | 1.72 |
| | Hydrogen | ......... | ......... | 5.31 | 5.04 | ......... | ......... | 5.09 |
| | Carbon | ......... | ......... | 59.72 | 50.22 | ......... | ......... | 65.48 |
| | Nitrogen | ......... | ......... | 1.03 | 0.72 | ......... | ......... | 1.39 |
| | Oxygen | ......... | ......... | 16.64 | 17.58 | ......... | ......... | 15.04 |
| Colorific value determined: | | | | | | | | |
| | Calories | 6,235 | ......... | 6,125 | 5,083 | 6,881 | ......... | 6,542 |
| | British thermal units | 11,223 | ......... | 11,025 | 9,149 | 12,386 | ......... | 11,776 |
| Calorific value calculated from ultimate analysis: | | | | | | | | |
| | Calories | ......... | ......... | 6,031 | 5,124 | ......... | ......... | 6,436 |
| | British thermal units | ......... | ......... | 10,856 | 9,223 | ......... | ......... | 11,585 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Illinois No. 4. | | | Illinois No. 5. | Illinois No. 6. | | |
|---|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| | Laboratory sample No. | 1341 | 1342 | 1417 | 1556 | 1449 | 1450 | 1557 |
| | Loss of moisture on air-drying. | 3.20 | 3.00 | 1.70 | 12.50 | 5.60 | 4.00 | 9.80 |
| | *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | 12.28 | 11.77 | 11.40 | 5.16 | 9.84 | 10.35 | 5.13 |
| | Volatile matter | 32.02 | 33.18 | 32.45 | 34.98 | 36.86 | 35.85 | 32.68 |
| | Fixed carbon | 38.03 | 45.97 | 44.30 | 40.67 | 44.96 | 42.94 | 47.46 |
| | Ash | 7.67 | 9.06 | 11.85 | 19.19 | 8.34 | 11.36 | 14.73 |
| Ultimate | Sulphur | 0.86 | 1.57 | 1.34 | 3.76 | 3.82 | 3.95 | 4.45 |
| | Hydrogen | .......... | .......... | 5.83 | 4.69 | .......... | .......... | 4.88 |
| | Carbon | .......... | .......... | 61.79 | 58.02 | .......... | .......... | 60.51 |
| | Nitrogen | .......... | .......... | 1.17 | .......... | .......... | .......... | 1.23 |
| | Oxygen | .......... | .......... | 18.52 | .......... | .......... | .......... | 14.20 |
| | Calorific value determined: | | | | | | | |
| | Calories | 6,400 | .......... | 6,106 | 5,917 | 6,483 | .......... | 6,199 |
| | British thermal units | 11,520 | .......... | 10,991 | 10,651 | 11,669 | .......... | 11,158 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 6,062 | .......... | .......... | .......... | 6,059 |
| | British thermal units | .......... | .......... | 10,912 | .......... | .......... | .......... | 10,906 |
| | *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 15.09 | 14.42 | 12.91 | 17.02 | 14.89 | 13.94 | 14.43 |
| | Volatile matter | 31.00 | 32.18 | 31.90 | 30.60 | 34.80 | 33.93 | 29.48 |
| | Fixed carbon | 46.49 | 44.59 | 43.55 | 35.59 | 42.44 | 41.22 | 42.81 |
| | Ash | 7.42 | 8.81 | 11.64 | 16.79 | 7.87 | 10.91 | 13.28 |
| Ultimate | Sulphur | 0.83 | 1.52 | 1.32 | 3.29 | 3.61 | 3.79 | 4.01 |
| | Hydrogen | .......... | .......... | 5.43 | 5.50 | .......... | .......... | 5.49 |
| | Carbon | .......... | .......... | 60.74 | 50.77 | .......... | .......... | 54.59 |
| | Nitrogen | .......... | .......... | 1.15 | .......... | .......... | .......... | 1.11 |
| | Oxygen | .......... | .......... | 19.72 | .......... | .......... | .......... | 21.52 |
| | Calorific value determined: | | | | | | | |
| | Calories | 6,195 | .......... | 6,002 | 5,177 | 6,120 | .......... | 5,591 |
| | British thermal units | 11,151 | .......... | 10,804 | 9,319 | 11,016 | .......... | 10,064 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 5,959 | .......... | .......... | .......... | 5,465 |
| | British thermal units | .......... | .......... | 10,726 | .......... | .......... | .......... | 9,837 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Indiana No. 1. | | | Indiana No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| | Laboratory sample No | 1410 | 1412 | 1507 | 1425 | 1426 | 1495 |
| | Loss of moisture on air drying | 8.70 | 5.60 | 3.00 | 2.80 | 3.10 | 3.60 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 4.98 | 6.25 | 8.66 | 6.67 | 7.45 | 6.24 |
| | Volatile matter | 39.22 | 37.10 | 34.86 | 40.53 | 39.30 | 37.49 |
| | Fixed carbon | 45.76 | 46.46 | 42.67 | 43.19 | 44.43 | 42.76 |
| | Ash | 10.04 | 10.19 | 13.81 | 9.61 | 8.82 | 13.51 |
| Ultimate | Sulphur | 2.05 | 1.02 | 2.58 | 4.57 | 3.62 | 4.60 |
| | Hydrogen | ........ | ........ | 5.20 | ........ | ........ | 5.11 |
| | Carbon | ........ | ........ | 62.20 | ........ | ........ | 62.97 |
| | Nitrogen | ........ | ........ | 1.22 | ........ | ........ | 1.25 |
| | Oxygen | ........ | ........ | 14.99 | ........ | ........ | 12.56 |
| | Calorific value determined: | | | | | | |
| | Calories | 6,912 | ........ | 6,336 | 6,744 | ........ | 6,410 |
| | British thermal units | 12,442 | ........ | 11,405 | 12,139 | ........ | 11,538 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 6,231 | ........ | ........ | 6,411 |
| | British thermal units | ........ | ........ | 11,216 | ........ | ........ | 11,540 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 13.25 | 11.50 | 11.40 | 9.28 | 10.32 | 9.62 |
| | Volatile matter | 35.81 | 35.02 | 33.81 | 39.40 | 38.08 | 36.14 |
| | Fixed carbon | 41.78 | 43.86 | 41.39 | 41.98 | 43.05 | 41.22 |
| | Ash | 9.16 | 9.62 | 13.40 | 9.34 | 8.55 | 13.02 |
| Ultimate | Sulphur | 1.87 | 0.96 | 2.50 | 4.44 | 3.51 | 4.43 |
| | Hydrogen | ........ | ........ | 5.37 | ........ | ........ | 5.33 |
| | Carbon | ........ | ........ | 60.34 | ........ | ........ | 60.70 |
| | Nitrogen | ........ | ........ | 1.18 | ........ | ........ | 1.20 |
| | Oxygen | ........ | ........ | 17.21 | ........ | ........ | 15.32 |
| | Calorific value determined: | | | | | | |
| | Calories | 6,311 | ........ | 6,145 | 6,555 | ........ | 6,179 |
| | British thermal units | 11,360 | ........ | 11,061 | 11,799 | ........ | 11,122 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 6,044 | ........ | ........ | 6,180 |
| | British thermal units | ........ | ........ | 10,879 | ........ | ........ | 11,124 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Indian Territory No. 1. | | | Indian Territory No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| Laboratory sample No | | 1059 | 1060 | 1138 | 1071 | 1073 | 1184 |
| Loss of moisture on air drying | | 5.00 | 3.10 | 3.30 | | | 2.80 |
| *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | 4.07 | 3.79 | 3.87 | | | 1.70 |
| | Volatile matter | 36.65 | 37.41 | 35.73 | | | 37.19 |
| | Fixed carbon | 50.20 | 52.94 | 50.05 | | | 49.79 |
| | Ash | 9.08 | 5.86 | 10.35 | | | 11.32 |
| Ultimate | Sulphur | 1.71 | 1.40 | 1.99 | | | 1.56 |
| | Hydrogen | | | 5.14 | | | 5.00 |
| | Carbon | | | 69.85 | | | 71.49 |
| | Nitrogen | | | 1.29 | | | 1.72 |
| | Oxygen | | | 11.38 | | | 8.91 |
| Calorific value determined: | | | | | | | |
| | Calories | 7,073 | | 7,011 | | | 7,205 |
| | British thermal units | 12,731 | | 12,620 | | | 12,969 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | | | 6,971 | | | 7,152 |
| | British thermal units | | | 12,548 | | | 12,874 |
| *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 8.87 | 6.77 | 7.04 | 1.46 | 1.30 | 4.45 |
| | Volatile matter | 34.82 | 36.25 | 34.55 | 39.04 | 38.90 | 36.15 |
| | Fixed carbon | 47.68 | 51.30 | 48.40 | 53.10 | 52.15 | 48.40 |
| | Ash | 8.63 | 5.68 | 10.01 | 6.40 | 7.65 | 11.00 |
| Ultimate | Sulphur | 1.62 | 1.36 | 1.92 | 1.38 | 1.58 | 1.52 |
| | Hydrogen | | | 5.34 | | | 5.17 |
| | Carbon | | | 67.55 | | | 69.49 |
| | Nitrogen | | | 1.25 | | | 1.67 |
| | Oxygen | | | 13.93 | | | 11.15 |
| Calorific value determined: | | | | | | | |
| | Calories | 6,720 | | 6,779 | 7,800 | | 7,004 |
| | British thermal units | 12,096 | | 12,202 | 14,040 | | 12,607 |
| Calorific value calculaled from ultimate analysis: | | | | | | | |
| | Calories | | | 6,740 | | | 6,952 |
| | British thermal units | | | 12,132 | | | 12,514 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Indian Territory No. 3. | | | Indian Territory No. 4. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Labora-tory car sample. | Mine sample A. | Mine sample B. | Labora-tory car sample. |
| | Laboratory sample No. | 1079 | 1080 | 1274 | 1150 | 1151 | 1470 |
| | Loss of moisture on air drying | | | 1.20 | 1.60 | 1.90 | 1.40 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | | | 3.45 | 3.75 | 4.69 | 4.91 |
| | Volatile matter | | | 37.45 | 38.15 | 39.76 | 37.79 |
| | Fixed carbon | | | 47.82 | 45.77 | 46.06 | 43.90 |
| | Ash | | | 11.28 | 12.33 | 9.49 | 13.40 |
| Ultimate | Sulphur | | | 3.67 | 3.83 | 3.74 | 4.02 |
| | Hydrogen | | | 4.85 | | | 4.84 |
| | Carbon | | | 68.18 | | | 63.21 |
| | Nitrogen | | | 1.50 | | | 1.38 |
| | Oxygen | | | 10.52 | | | 13.15 |
| | Calorific value determined: | | | | | | |
| | Calories | | | 6,927 | | 6,706 | 6,327 |
| | British thermal units | | | 12,469 | | 12,071 | 11,389 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 6,810 | | | 6,300 |
| | British thermal units | | | 12,258 | | | 11,340 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 2.97 | 2.93 | 4.61 | 5.29 | 6.50 | 6.24 |
| | Volatile matter | 40.43 | 39.02 | 37.00 | 37.54 | 39.01 | 37.26 |
| | Fixed carbon | 48.22 | 47.75 | 47.25 | 45.04 | 45.18 | 43.29 |
| | Ash | 8.38 | 10.30 | 11.14 | 12.13 | 9.31 | 13.21 |
| Ultimate | Sulphur | 3.05 | 3.73 | 3.63 | 3.77 | 3.67 | 3.96 |
| | Hydrogen | | | 4.92 | | | 4.93 |
| | Carbon | | | 67.37 | | | 62.34 |
| | Nitrogen | | | 1.48 | | | 1.36 |
| | Oxygen | | | 11.46 | | | 14.20 |
| | Calorific value determined: | | | | | | |
| | Calories | | 6,995 | 6,844 | | 6,579 | 6,238 |
| | British thermal units | | 12,591 | 12,319 | | 11,842 | 11,228 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 6,728 | | | 6,212 |
| | British thermal units | | | 12,110 | | | 11,182 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Ind. Ter. No. 5. | Ind. Ter. No. 6. | Iowa No. 1. | | |
|---|---|---|---|---|---|---|
| | | Laboratory car sample. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| Laboratory sample No. | | 1481 | 1596 | 1270 | 1271 | 1347 |
| Loss of moisture on air drying | | 2.70 | 3.50 | 7.90 | 8.00 | 3.20 |
| *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 5.74 | 4.69 | 3.74 | 4.43 | 5.21 |
| | Volatile matter | 31.46 | 32.41 | 41.96 | 40.52 | 31.76 |
| | Fixed carbon | 37.05 | 42.91 | 42.89 | 41.65 | 46.51 |
| | Ash | 25.75 | 19.99 | 11.41 | 13.40 | 16.52 |
| Ultimate | Sulphur | 4.06 | 3.32 | 5.12 | 5.42 | 5.20 |
| | Hydrogen | 4.18 | ... | ... | ... | 4.61 |
| | Carbon | 52.39 | ... | ... | ... | 61.80 |
| | Nitrogen | 1.22 | ... | ... | ... | 0.97 |
| | Oxygen | 12.40 | ... | ... | ... | 10.90 |
| Calorific value determined: | | | | | | |
| | Calories | 5,201 | ... | 6,843 | ... | 6,329 |
| | British thermal units | 9,362 | ... | 12,317 | ... | 11,392 |
| Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | 5,231 | ... | ... | ... | 6,230 |
| | British thermal units | 9,416 | ... | ... | ... | 11,214 |
| *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 8.29 | 8.03 | 11.35 | 12.07 | 8.24 |
| | Volatile matter | 30.61 | 31.28 | 38.65 | 37.28 | 30.74 |
| | Fixed carbon | 36.05 | 41.40 | 39.49 | 38.32 | 45.02 |
| | Ash | 25.05 | 19.29 | 10.51 | 12.33 | 16.00 |
| Ultimate | Sulphur | 3.95 | 3.20 | 4.72 | 4.99 | 5.03 |
| | Hydrogen | 4.37 | ... | ... | ... | 4.81 |
| | Carbon | 50.98 | ... | ... | ... | 59.82 |
| | Nitrogen | 1.19 | ... | ... | ... | 0.94 |
| | Oxygen | 14.46 | ... | ... | ... | 13.40 |
| Calorific value determined: | | | | | | |
| | Calories | 5,061 | ... | 6,303 | ... | 6,126 |
| | British thermal units | 9,110 | ... | 11,345 | ... | 11,027 |
| Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | 5,090 | ... | ... | ... | 6,031 |
| | British thermal units | 9,162 | ... | ... | ... | 10,856 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Iowa No. 2. | | | Iowa No. 3. | | |
|---|---|---|---|---|---|---|---|
| | | **Mine sample B.** | **Mine sample A.** | **Laboratory car sample.** | **Mine sample B.** | **Mine sample A.** | **Laboratory car sample.** |
| | Laboratory sample No | 1289 | 1291 | 1570 | 1312 | 1313 | 1434 |
| | Loss of moisture on air drying | 9.30 | 9.50 | 10.40 | 9.60 | 11.00 | 9.80 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 7.00 | 6.63 | 4.25 | 5.33 | 5.51 | 4.52 |
| | Volatile matter | 40.65 | 40.82 | 37.02 | 41.82 | 42.04 | 40.96 |
| | Fixed carbon | 39.52 | 42.40 | 41.74 | 40.69 | 38.55 | 38.99 |
| | Ash | 12.83 | 10.15 | 16.99 | 12.16 | 13.90 | 15.53 |
| Ultimate | Sulphur | 5.49 | 5.74 | 5.20 | 6.52 | 7.59 | 6.83 |
| | Hydrogen | .......... | .......... | 4.84 | .......... | .......... | 4.93 |
| | Carbon | .......... | .......... | 60.36 | .......... | .......... | 60.62 |
| | Nitrogen | .......... | .......... | 1.46 | .......... | .......... | 0.93 |
| | Oxygen | .......... | .......... | 11.15 | .......... | .......... | 11.16 |
| | Calorific value determined: | | | | | | |
| | Calories | 6,302 | .......... | 6,212 | 6,539 | .......... | 6,309 |
| | British thermal units | 11,344 | .......... | 11,182 | 11,770 | .......... | 11,356 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | .......... | .......... | 6,188 | .......... | .......... | 6,271 |
| | British thermal units | .......... | .......... | 11,129 | .......... | .......... | 11,288 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 15.65 | 15.50 | 14.21 | 14.42 | 15.90 | 13.88 |
| | Volatile matter | 36.87 | 36.94 | 33.17 | 37.81 | 37.42 | 36.94 |
| | Fixed carbon | 35.84 | 38.37 | 37.40 | 36.78 | 34.41 | 35.17 |
| | Ash | 11.64 | 9.19 | 15.22 | 10.99 | 12.37 | 14.01 |
| Ultimate | Sulphur | 5.10 | 5.19 | 4.66 | 5.89 | 6.76 | 6.15 |
| | Hydrogen | .......... | .......... | 5.50 | .......... | .......... | 5.52 |
| | Carbon | .......... | .......... | 54.08 | .......... | .......... | 54.68 |
| | Nitrogen | .......... | .......... | 1.31 | .......... | .......... | 0.84 |
| | Oxygen | .......... | .......... | 19.23 | .......... | .......... | 18.80 |
| | Calorific value determined: | | | | | | |
| | Calories | 5,716 | .......... | 5,566 | 5,911 | .......... | 5,691 |
| | British thermal units | 10,289 | .......... | 10,019 | 10,640 | .......... | 10,244 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | .......... | .......... | 5,540 | .......... | .......... | 5,656 |
| | British thermal units | .......... | .......... | 9,972 | .......... | .......... | 10,181 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Iowa No. 4. | | | Iowa No. 5. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| | Laboratory sample No. | 1323 | 1324 | 1437 | 1332 | 1333 | 1433 |
| | Loss of moisture on air drying | 9.40 | 8.60 | 4.50 | 9.40 | 7.10 | 6.80 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 8.53 | 8.25 | 10.03 | 10.25 | 12.37 | 9.22 |
| | Volatile matter | 39.12 | 38.23 | 37.27 | 35.10 | 36.98 | 32.71 |
| | Fixed carbon | 44.55 | 41.40 | 41.22 | 46.12 | 42.95 | 44.52 |
| | Ash | 7.80 | 12.12 | 11.48 | 8.53 | 7.70 | 13.55 |
| Ultimate | Sulphur | 4.42 | 5.21 | 4.46 | 2.64 | 3.34 | 3.42 |
| | Hydrogen | ........ | ........ | 5.31 | ........ | ........ | 5.35 |
| | Carbon | ........ | ........ | 61.25 | ........ | ........ | 59.89 |
| | Nitrogen | ........ | ........ | 0.94 | ........ | ........ | 1.22 |
| | Oxygen | ........ | ........ | 16.56 | ........ | ........ | 16.57 |
| | Calorific value determined: | | | | | | |
| | Calories | 6,703 | ........ | 6,237 | 6,442 | ........ | 6,105 |
| | British thermal units | 12,065 | ........ | 11,227 | 11,596 | ........ | 10.989 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 6,165 | ........ | ........ | 6,045 |
| | British thermal units | ........ | ........ | 11,097 | ........ | ........ | 10,881 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 17.13 | 16.14 | 14.08 | 18.69 | 18.59 | 15.39 |
| | Volatile matter | 35.44 | 34.94 | 35.59 | 31.80 | 34.36 | 30.49 |
| | Fixed carbon | 40.36 | 37.84 | 39.37 | 41.78 | 39.90 | 41.49 |
| | Ash | 7.07 | 11.08 | 10.96 | 7.73 | 7.15 | 12.63 |
| Ultimate | Sulphur | 4.00 | 4.76 | 4.26 | 2.39 | 3.10 | 3.19 |
| | Hydrogen | ........ | ........ | 5.57 | ........ | ........ | 5.74 |
| | Carbon | ........ | ........ | 58.49 | ........ | ........ | 55.81 |
| | Nitrogen | ........ | ........ | 0.90 | ........ | ........ | 1.14 |
| | Oxygen | ........ | ........ | 19.82 | ........ | ........ | 21.49 |
| | Calorific value determined: | | | | | | |
| | Calories | 6,073 | ........ | 5,957 | 5,836 | ........ | 5,690 |
| | British thermal units | 10,931 | ........ | 10,723 | 10,506 | ........ | 10,242 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 5,888 | ........ | ........ | 5,634 |
| | British thermal units | ........ | ........ | 10,598 | ........ | ........ | 10,141 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Kansas No. 1. | | | Kansas No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| Laboratory sample No | | 1018 | 1020 | 1097 | 1017 | 1019 | 1122 |
| Loss of moisture on air drying | | .......... | .......... | 1.30 | .......... | .......... | 2.00 |
| *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | .......... | .......... | 3.74 | .......... | .......... | 2.23 |
| | Volatile matter | .......... | .......... | 33.11 | .......... | .......... | 31.87 |
| | Fixed carbon | .......... | .......... | 50.01 | .......... | .......... | 47.63 |
| | Ash | .......... | .......... | 13.14 | .......... | .......... | 18.27 |
| Ultimate | Sulphur | .......... | .......... | 4.34 | .......... | .......... | 6.40 |
| | Hydrogen | .......... | .......... | 4.91 | .......... | .......... | 4.56 |
| | Carbon | .......... | .......... | 68.22 | .......... | .......... | 63.14 |
| | Nitrogen | .......... | .......... | 1.09 | .......... | .......... | 0.94 |
| | Oxygen | .......... | .......... | 8.30 | .......... | .......... | 6.69 |
| Calorific value determined: | | | | | | | |
| | Calories | .......... | .......... | 6,891 | .......... | .......... | 6,600 |
| | British thermal units | .......... | .......... | 12,404 | .......... | .......... | 11,880 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 6,940 | .......... | .......... | 6,528 |
| | British thermal units | .......... | .......... | 12,492 | .......... | .......... | 11,750 |
| *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 2.91 | 3.50 | 4.99 | 2.44 | 2.36 | 4.18 |
| | Volatile matter | 35.81 | 35.75 | 32.68 | 35.16 | 34.62 | 31.23 |
| | Fixed carbon | 51.73 | 52.83 | 49.36 | 51.80 | 51.23 | 46.68 |
| | Ash | 9.55 | 7.92 | 12.97 | 10.60 | 11.79 | 17.91 |
| Ultimate | Sulphur | 3.79 | 3.28 | 4.28 | 5.63 | 5.88 | 6.27 |
| | Hydrogen | .......... | .......... | 4.98 | .......... | .......... | 4.69 |
| | Carbon | .......... | .......... | 67.34 | .......... | .......... | 61.88 |
| | Nitrogen | .......... | .......... | 1.08 | .......... | .......... | 0.92 |
| | Oxygen | .......... | .......... | 9.35 | .......... | .......... | 8.33 |
| Calorific value determined: | | | | | | | |
| | Calories | 7,193 | .......... | 6,801 | 7,246 | .......... | 6,468 |
| | British thermal units | 12,947 | .......... | 12,242 | 13,043 | .......... | 11,642 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 6,850 | .......... | .......... | 6,397 |
| | British thermal units | .......... | .......... | 12,330 | .......... | .......... | 11,515 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Kansas No. 3. | | | Kansas No. 4. | Kansas No. 5. | | |
|---|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| | Laboratory sample No. | 1036 | 1037 | 1086 | 1473 | 1411 | 1413 | 1567 |
| | Loss of moisture on air-drying | ......... | ......... | ......... | 3.50 | 3.20 | 4.30 | 2.30 |
| | *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | ......... | ......... | ......... | 3.57 | 1.97 | 1.56 | 1.84 |
| | Volatile matter | ......... | ......... | ......... | 37.00 | 33.68 | 33.79 | 32.40 |
| | Fixed carbon | ......... | ......... | ......... | 46.80 | 55.15 | 51.54 | 54.97 |
| | Ash | ......... | ......... | ......... | 12.63 | 9.20 | 13.11 | 10.79 |
| Ultimate | Sulphur | ......... | ......... | ......... | 8.33 | 4.48 | 4.01 | 3.86 |
| | Hydrogen | ......... | ......... | ......... | 5.04 | ......... | ......... | 4.96 |
| | Carbon | ......... | ......... | ......... | 65.02 | ......... | ......... | 71.90 |
| | Nitrogen | ......... | ......... | ......... | 1.07 | ......... | ......... | 1.09 |
| | Oxygen | ......... | ......... | ......... | 7.91 | ......... | ......... | 7.40 |
| | Calorific value determined: | | | | | | | |
| | Calories | ......... | ......... | ......... | 6,854 | 7,418 | ......... | 7,333 |
| | British thermal units | ......... | ......... | ......... | 12,337 | 13,352 | ......... | 13,199 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | ......... | ......... | ......... | 6,839 | ......... | ......... | 7,288 |
| | British thermal units | ......... | ......... | ......... | 12,310 | ......... | ......... | 13,118 |
| | *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 2.01 | 2.54 | 2.50 | 6.95 | 5.11 | 5.79 | 4.10 |
| | Volatile matter | 35.99 | 35.31 | 33.80 | 35.70 | 32.60 | 32.34 | 31.65 |
| | Fixed carbon | 46.85 | 52.28 | 51.25 | 45.16 | 53.39 | 49.32 | 53.71 |
| | Ash | 15.15 | 9.87 | 12.45 | 12.19 | 8.90 | 12.55 | 10.54 |
| Ultimate | Sulphur | 5.27 | 4.47 | 5.68 | 8.04 | 4.34 | 3.84 | 3.77 |
| | Hydrogen | ......... | ......... | 4.91 | 5.25 | ......... | ......... | 5.10 |
| | Carbon | ......... | ......... | 69.07 | 62.74 | ......... | ......... | 70.25 |
| | Nitrogen | ......... | ......... | 1.20 | 1.04 | ......... | ......... | 1.06 |
| | Oxygen | ......... | ......... | 6.69 | 10.74 | ......... | ......... | 9.28 |
| | Calorific value determined: | | | | | | | |
| | Calories | ......... | 7,411 | 7,166 | 6,614 | 7,181 | ......... | 7,164 |
| | British thermal units | ......... | 13,340 | 12,900 | 11,905 | 12,926 | ......... | 12,895 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | ......... | ......... | 7,111 | 6,600 | ......... | ......... | 7,121 |
| | British thermal units | ......... | ......... | 12,800 | 11,880 | ......... | ......... | 12,818 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Kentucky No. 1. | | | Kentucky No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| | Laboratory sample No | 1321 | 1322 | 1474 | 1365 | 1366 | 1461 |
| | Loss of moisture on air drying | 1.00 | 0.90 | 1.20 | 2.90 | 2.60 | 2.70 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 1.93 | 1.93 | 1.92 | 5.76 | 5.34 | 5.36 |
| Proximate | Volatile matter | 36.37 | 37.42 | 36.56 | 39.19 | 38.61 | 38.99 |
| Proximate | Fixed carbon | 58.13 | 57.83 | 57.08 | 47.74 | 45.56 | 46.27 |
| Proximate / Ultimate | Ash | 3.57 | 2.82 | 4.44 | 7.31 | 10.49 | 9.38 |
| Ultimate | Sulphur | 0.90 | 0.85 | 1.24 | 3.63 | 4.31 | 3.72 |
| Ultimate | Hydrogen | | | 5.36 | | | 5.33 |
| Ultimate | Carbon | | | 78.31 | | | 67.64 |
| Ultimate | Nitrogen | | | 1.85 | | | 1.25 |
| Ultimate | Oxygen | | | 8.80 | | | 12.68 |
| | Calorific value determined: | | | | | | |
| | Calories | 8,037 | | 7,955 | 7,063 | | 6,966 |
| | British thermal units | 14,467 | | 14,319 | 12,713 | | 12,539 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 7,823 | | | 6,840 |
| | British thermal units | | | 14,081 | | | 12,312 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 2.91 | 2.81 | 3.10 | 8.49 | 7.80 | 7.91 |
| Proximate | Volatile matter | 36.01 | 37.08 | 36.12 | 38.05 | 37.60 | 37.94 |
| Proximate | Fixed carbon | 57.55 | 57.31 | 56.39 | 46.36 | 44.38 | 45.02 |
| Proximate / Ultimate | Ash | 3.53 | 2.80 | 4.39 | 7.10 | 10.22 | 9.13 |
| Ultimate | Sulphur | 0.89 | 0.84 | 1.22 | 3.53 | 4.20 | 3.62 |
| Ultimate | Hydrogen | | | 5.43 | | | 5.48 |
| Ultimate | Carbon | | | 77.37 | | | 65.81 |
| Ultimate | Nitrogen | | | 1.83 | | | 1.22 |
| Ultimate | Oxygen | | | 9.76 | | | 14.74 |
| | Calorific value determined: | | | | | | |
| | Calories | 7,957 | | 7,860 | 6,858 | | 6,778 |
| | British thermal units | 14,322 | | 14,148 | 12,344 | | 12,200 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 7,729 | | | 6,655 |
| | British thermal units | | | 13,912 | | | 11,979 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Kentucky No. 3. | | | Kentucky No. 4. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| | Laboratory sample No | 1361 | 1367 | 1506 | 1382 | 1384 | 1539 |
| | Loss of moisture on air drying | 3.30 | 2.20 | 2.20 | 2.60 | 2.00 | 2.80 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 6.00 | 5.91 | 5.85 | 2.06 | 2.82 | 2.54 |
| Proximate | Volatile matter | 37.45 | 38.39 | 36.90 | 39.19 | 40.58 | 36.08 |
| Proximate | Fixed carbon | 48.23 | 46.19 | 46.96 | 51.15 | 49.50 | 46.79 |
| Proximate / Ultimate | Ash | 8.32 | 9.51 | 10.29 | 7.60 | 7.15 | 14.59 |
| Proximate / Ultimate | Sulphur | 3.07 | 4.12 | 3.60 | 3.42 | 3.28 | 4.67 |
| Ultimate | Hydrogen | ........ | ........ | 5.27 | ........ | ........ | 4.53 |
| Ultimate | Carbon | ........ | ........ | 66.75 | ........ | ........ | 66.50 |
| Ultimate | Nitrogen | ........ | ........ | 1.43 | ........ | ........ | 1.28 |
| Ultimate | Oxygen | ........ | ........ | 12.66 | ........ | ........ | 8.43 |
| | Calorific value determined: | | | | | | |
| | Calories | ........ | 6,797 | 6,829 | 7,336 | ........ | 6,830 |
| | British thermal units | ........ | 12,235 | 12,292 | 13,205 | ........ | 12,294 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 6,746 | ........ | ........ | 6,677 |
| | British thermal units | ........ | ........ | 12,143 | ........ | ........ | 12,019 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 9.10 | 7.98 | 7.92 | 4.61 | 4.76 | 5.27 |
| Proximate | Volatile matter | 36.21 | 37.55 | 36.09 | 38.17 | 39.72 | 35.07 |
| Proximate | Fixed carbon | 46.64 | 45.17 | 45.93 | 49.82 | 48.51 | 45.48 |
| Proximate / Ultimate | Ash | 8.05 | 9.30 | 10.06 | 7.40 | 7.01 | 14.18 |
| Proximate / Ultimate | Sulphur | 2.97 | 4.03 | 3.52 | 3.33 | 3.21 | 4.54 |
| Ultimate | Hydrogen | ........ | ........ | 5.39 | ........ | ........ | 4.71 |
| Ultimate | Carbon | ........ | ........ | 65.29 | ........ | ........ | 64.65 |
| Ultimate | Nitrogen | ........ | ........ | 1.40 | ........ | ........ | 1.24 |
| Ultimate | Oxygen | ........ | ........ | 14.34 | ........ | ........ | 10.68 |
| | Calorific value determined: | | | | | | |
| | Calories | ........ | 6,647 | 6,679 | 7,145 | ........ | 6,639 |
| | British thermal units | ........ | 11,965 | 12,022 | 12,861 | ........ | 11,950 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | ........ | ........ | 6,598 | ........ | ........ | 6,490 |
| | British thermal units | ........ | ........ | 11,876 | ........ | ........ | 11,682 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Missouri No. 1. | | | Missouri No. 2. | | | Missouri No. 3. |
|---|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. | Laboratory car sample. |
| | Laboratory sample No | 1041 | 1043 | 1126 | 1226 | 1227 | 1348 | 1549 |
| | Loss of moisture on air-drying | .......... | .......... | 5.00 | 6.20 | 5.00 | 2.60 | 10.80 |
| | *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | .......... | .......... | 3.50 | 9.10 | 8.31 | 9.14 | 5.51 |
| | Volatile matter | .......... | .......... | 35.35 | 41.07 | 38.47 | 34.53 | 32.08 |
| | Fixed carbon | .......... | .......... | 40.77 | 41.53 | 42.00 | 39.02 | 39.11 |
| | Ash | .......... | .......... | 20.38 | 8.30 | 11.22 | 17.31 | 23.30 |
| Ultimate | Sulphur | .......... | .......... | 5.53 | 4.04 | 4.03 | 5.30 | 4.13 |
| | Hydrogen | .......... | .......... | 4.64 | .......... | .......... | 4.96 | 4.52 |
| | Carbon | .......... | .......... | 60.00 | .......... | .......... | 56.25 | 54.79 |
| | Nitrogen | .......... | .......... | 0.99 | .......... | .......... | 0.99 | 0.92 |
| | Oxygen | .......... | .......... | 8.46 | .......... | .......... | 15.19 | 12.34 |
| | Calorific value determined: | | | | | | | |
| | Calories | .......... | .......... | 6,191 | 6,625 | .......... | 5,806 | 5,506 |
| | British thermal units | .......... | .......... | 11,144 | 11,925 | .......... | 10,451 | 9,911 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 6,206 | .......... | .......... | 5,719 | 5,547 |
| | British thermal units | .......... | .......... | 11,171 | .......... | .......... | 10,294 | 9,985 |
| | *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 4.80 | 4.92 | 8.33 | 14.74 | 12.90 | 11.50 | 15.71 |
| | Volatile matter | 38.10 | 38.28 | 33.58 | 38.53 | 36.54 | 33.63 | 28.62 |
| | Fixed carbon | 42.93 | 42.28 | 38.73 | 38.95 | 39.90 | 38.01 | 34.89 |
| | Ash | 14.17 | 14.52 | 19.36 | 7.78 | 10.66 | 16.86 | 20.78 |
| Ultimate | Sulphur | 5.35 | 5.34 | 5.25 | 3.79 | 3.83 | 5.16 | 3.69 |
| | Hydrogen | .......... | .......... | 4.97 | .......... | .......... | 5.12 | 5.23 |
| | Carbon | .......... | .......... | 57.00 | .......... | .......... | 54.79 | 48.87 |
| | Nitrogen | .......... | .......... | 0.94 | .......... | .......... | 0.96 | 0.82 |
| | Oxygen | .......... | .......... | 12.48 | .......... | .......... | 17.11 | 20.61 |
| | Calorific value determined: | | | | | | | |
| | Calories | .......... | 6,662 | 5,881 | 6,214 | .......... | 5,655 | 4,911 |
| | British thermal units | .......... | 11,992 | 10,586 | 11,185 | .......... | 10,179 | 8,840 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 5,896 | .......... | .......... | 5,570 | 4,948 |
| | British thermal units | .......... | .......... | 10,613 | .......... | .......... | 10,026 | 8,906 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Missouri No. 4. | | | Montana No. 1. | New Mexico No. 1. | | |
|---|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Laboratory car sample. | Mine samples A & B.[a] | Mine samples C & D.[a] | Laboratory car sample. |
| | Laboratory sample No. | 1446 | 1447 | 1516 | 1298 | 1023<br>1024 | 1025<br>1026 | 1278 |
| | Loss of moisture on air drying. | 7.40 | 6.00 | 7.70 | 2.20 | .......... | .......... | 1.60 |
| | *Analysis of air-dried samples.* | | | | | | | |
| Proximate | Moisture | 6.42 | 4.86 | 5.39 | 9.05 | .......... | .......... | 10.86 |
| | Volatile matter | 40.73 | 43.74 | 44.91 | 36.70 | .......... | .......... | 35.14 |
| | Fixed carbon | 45.39 | 44.86 | 44.47 | 43.03 | .......... | .......... | 46.90 |
| | Ash | 7.46 | 6.54 | 5.23 | 11.22 | .......... | .......... | 7.10 |
| Ultimate | Sulphur | 5.46 | 5.32 | 5.55 | 1.76 | .......... | .......... | 0.64 |
| | Hydrogen | .......... | .......... | 5.77 | 5.25 | .......... | .......... | 5.73 |
| | Carbon | .......... | .......... | 72.45 | 60.41 | .......... | .......... | 64.34 |
| | Nitrogen | .......... | .......... | 0.75 | 1.36 | .......... | .......... | 1.05 |
| | Oxygen | .......... | .......... | 10.25 | 20.00 | .......... | .......... | 21.14 |
| | Calorific value determined: | | | | | | | |
| | Calories | 6,962 | .......... | 7,516 | 5,987 | .......... | .......... | 6,353 |
| | British thermal units | 12,582 | .......... | 13,529 | 10,777 | .......... | .......... | 11,435 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 7,526 | 5,868 | .......... | .......... | 6,277 |
| | British thermal units | .......... | .......... | 13,547 | 10,562 | .......... | .......... | 11,299 |
| | *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 13.34 | 10.57 | 12.67 | 11.05 | 11.77 | 10.96 | 12.29 |
| | Volatile matter | 87.72 | 41.11 | 41.45 | 35.90 | 41.85 | 42.63 | 34.58 |
| | Fixed carbon | 42.03 | 42.17 | 41.05 | 42.08 | 43.11 | 42.39 | 46.14 |
| | Ash | 6.91 | 6.15 | 4.83 | 10.97 | 3.26 | 4.01 | 6.99 |
| Ultimate | Sulphur | 5.06 | 5.00 | 5.12 | 1.78 | 0.54 | 0.52 | 0.63 |
| | Hydrogen | .......... | .......... | 6.18 | 5.37 | .......... | .......... | 5.82 |
| | Carbon | .......... | .......... | 66.87 | 59.08 | .......... | .......... | 63.31 |
| | Nitrogen | .......... | .......... | 0.69 | 1.83 | .......... | .......... | 1.03 |
| | Oxygen | .......... | .......... | 16.31 | 21.52 | .......... | .......... | 22.22 |
| | Calorific value determined: | | | | | | | |
| | Calories | 6,447 | .......... | 6,937 | 5,855 | .......... | 6,608 | 6,251 |
| | British thermal units | 11,605 | .......... | 12,487 | 10,539 | .......... | 11,885 | 11,252 |
| | Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 6,946 | 5,739 | .......... | .......... | 6,177 |
| | British thermal units | .......... | .......... | 12,503 | 10,330 | .......... | .......... | 11,119 |

[a] Average of two samples.

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | New Mexico No. 2. | | | | |
|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Mine sample C. | Mine sample D. | Laboratory car sample. |
| | Laboratory sample No. | 1027 | 1028 | 1029 | 1038 | 1307 |
| | Loss of moisture on air drying | | | | | 2.90 |
| | *Analysis of air-dried sample.* | | | | | |
| Proximate | Moisture | | | | | 8.13 |
| | Volatile matter | | | | | 34.82 |
| | Fixed carbon | | | | | 37.83 |
| | Ash | | | | | 19.22 |
| Ultimate | Sulphur | | | | | 1.30 |
| | Hydrogen | | | | | 5.05 |
| | Carbon | | | | | 56.71 |
| | Nitrogen | | | | | 0.98 |
| | Oxygen | | | | | 16.74 |
| | Calorific value determined: | | | | | |
| | Calories | | | | | 5,668 |
| | British thermal units | | | | | 10,202 |
| | Calorific value calculated from ultimate analysis: | | | | | |
| | Calories | | | | | 5,631 |
| | British thermal units | | | | | 10,136 |
| | *Analysis corrected to sample as received.* | | | | | |
| Proximate | Moisture | 9.13 | 9.68 | 9.40 | 10.80 | 10.79 |
| | Volatile matter | 40.77 | 41.42 | 40.05 | 40.35 | 33.82 |
| | Fixed carbon | 40.23 | 40.82 | 37.87 | 42.77 | 36.73 |
| | Ash | 9.87 | 8.08 | 12.68 | 6.08 | 18.66 |
| Ultimate | Sulphur | 1.27 | 1.55 | 0.84 | 1.06 | 1.26 |
| | Hydrogen | | | | | 5.22 |
| | Carbon | | | | | 55.07 |
| | Nitrogen | | | | | 0.95 |
| | Oxygen | | | | | 18.84 |
| | Calorific value determined: | | | | | |
| | Calories | | 6,457 | | | 5,504 |
| | British thermal units | | 11,623 | | | 9,907 |
| | Calorific value calculated from ultimate analysis: | | | | | |
| | Calories | | | | | 5,468 |
| | British thermal units | | | | | 9,842 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | North Dakota No. 1. | North Dakota No. 2. | Pennsylvania No. 1. | Pennsylvania No. 2. | Pennsylvania No. 3. |
|---|---|---|---|---|---|---|
| | | Laboratory car sample. | Laboratory car sample. | Steam sample. | Steam sample. | Laboratory car sample. |
| **Laboratory sample No** | | 1279 | 1416 | ........ | ........ | 1245 |
| **Loss of moisture on air-drying** | | 23.60 | 24.10 | ........ | ........ | 3.40 |
| ***Analysis of air-dried sample.*** | | | | | | |
| Proximate | **Moisture** | 15.42 | 16.70 | ........ | ........ | 2.08 |
| | **Volatile matter** | 38.73 | 37.10 | ........ | ........ | 7.27 |
| | **Fixed carbon** | 33.61 | 39.49 | ........ | ........ | 74.32 |
| Proximate / Ultimate | **Ash** | 12.24 | 6.71 | ........ | ........ | 16.33 |
| | **Sulphur** | 2.02 | 0.63 | ........ | ........ | 0.77 |
| Ultimate | **Hydrogen** | 5.22 | 5.61 | ........ | ........ | 2.81 |
| | **Carbon** | 52.66 | 55.16 | ........ | ........ | 75.21 |
| | **Nitrogen** | 0.71 | 0.91 | ........ | ........ | 0.80 |
| | **Oxygen** | 27.15 | 30.98 | ........ | ........ | 4.08 |
| **Calorific value determined:** | | | | | | |
| | **Calories** | 5,084 | 5,278 | ........ | ........ | 6,929 |
| | **British thermal units** | 9.061 | 9,491 | ........ | ........ | 12,472 |
| **Calorific value calculated from ultimate analysis:** | | | | | | |
| | **Calories** | 4,931 | 5,071 | ........ | ........ | 6,886 |
| | **British thermal units** | 8,876 | 9,128 | ........ | ........ | 12,395 |
| ***Analysis corrected to sample as received.*** | | | | | | |
| Proximate | **Moisture** | 35.38 | 36.78 | 1.10 | 0.59 | 5.41 |
| | **Volatile matter** | 29.59 | 28.16 | 15.80 | 16.61 | 7.02 |
| | **Fixed carbon** | 25.68 | 29.97 | 75.69 | 76.76 | 71.79 |
| Proximate / Ultimate | **Ash** | 9.35 | 5.09 | 7.41 | 6.04 | 15.78 |
| | **Sulphur** | 1.55 | 0.48 | 1.49 | 0.91 | 0.74 |
| Ultimate | **Hydrogen** | 6.61 | 6.93 | ........ | ........ | 3.10 |
| | **Carbon** | 40.23 | 41.87 | ........ | ........ | 72.65 |
| | **Nitrogen** | 0.54 | 0.69 | ........ | ........ | 0.77 |
| | **Oxygen** | 41.72 | 44.94 | ........ | ........ | 6.96 |
| **Calorific value determined:** | | | | | | |
| | **Calories** | 3,846 | 4,002 | ........ | ........ | 6,693 |
| | **British thermal units** | 6,923 | 7,204 | ........ | ........ | 12,047 |
| **Calorific value calculated from ultimate analysis:** | | | | | | |
| | **Calories** | 3,769 | 3,849 | ........ | ........ | 6,652 |
| | **British thermal units** | 6,784 | 6,928 | ........ | ........ | 11,974 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Texas No. 1. | | | Texas No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample $B_1$ | Mine sample A. | Laboratory car sample. | Mine sample A. | Mine sample B. | Gas-producer sample. |
| Laboratory sample No | | 1195 | 1196 | 1456 | 1241 | 1243 | 1597 |
| Loss of moisture on air drying | | 14.40 | 16.30 | 24.60 | 5.80 | 5.90 | 25.80 |
| *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | 21.25 | 20.55 | 13.40 | 24.48 | 27.04 | 10.66 |
| | Volatile matter | 43.25 | 47.20 | 42.75 | 38.17 | 43.76 | 39.42 |
| | Fixed carbon | 22.85 | 19.41 | 29.00 | 28.94 | 20.17 | 40.11 |
| | Ash | 12.65 | 12.84 | 14.85 | 8.41 | 9.03 | 9.81 |
| Ultimate | Sulphur | 0.65 | 0.67 | 1.04 | 0.53 | 0.61 | 0.71 |
| | Hydrogen | .......... | .......... | 5.57 | .......... | .......... | 5.28 |
| | Carbon | .......... | .......... | 52.06 | .......... | .......... | 57.31 |
| | Nitrogen | .......... | .......... | 0.95 | .......... | .......... | 1.06 |
| | Oxygen | .......... | .......... | 25.53 | .......... | .......... | 25.83 |
| Calorific value determined: | | | | | | | |
| | Calories | .......... | 4,741 | 5,199 | 4,716 | .......... | 5,502 |
| | British thermal units | .......... | 8,534 | 9,358 | 8,489 | .......... | 9,904 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 5,046 | .......... | .......... | 5,352 |
| | British thermal units | .......... | .......... | 9,083 | .......... | .......... | 9,634 |
| *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 32.58 | 33.50 | 34.70 | 28.86 | 31.34 | 33.71 |
| | Volatile matter | 37.02 | 39.50 | 32.23 | 35.96 | 41.18 | 29.25 |
| | Fixed carbon | 19.56 | 16.25 | 21.87 | 27.26 | 18.98 | 29.76 |
| | Ash | 10.84 | 10.75 | 11.20 | 7.92 | 8.50 | 7.28 |
| Ultimate | Sulphur | 0.56 | 0.56 | 0.79 | 0.50 | 0.57 | 0.53 |
| | Hydrogen | .......... | .......... | 6.93 | .......... | .......... | 6.79 |
| | Carbon | .......... | .......... | 39.25 | .......... | .......... | 42.52 |
| | Nitrogen | .......... | .......... | 0.72 | .......... | .......... | 0.79 |
| | Oxygen | .......... | .......... | 41.11 | .......... | .......... | 42.09 |
| Calorific value determined: | | | | | | | |
| | Calories | .......... | 3,968 | 3,920 | 4,442 | .......... | 4,082 |
| | British thermal units | .......... | 7,142 | 7,056 | 7,996 | .......... | 7,348 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 3,805 | .......... | .......... | 3,975 |
| | British thermal units | .......... | .......... | 6,849 | .......... | .......... | 7,155 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | West Virginia No. 1. | | | West Virginia No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| | Laboratory sample No | 1088 | 1089 | 1213 | 1103 | 1104 | 1308 |
| | Loss of moisture on air drying | 0.40 | 0.40 | 0.40 | 0.90 | 0.80 | 0.50 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 1.00 | 0.95 | 1.35 | 1.09 | 1.08 | 1.46 |
| | Volatile matter | 36.80 | 37.50 | 36.92 | 40.91 | 40.62 | 40.14 |
| | Fixed carbon | 55.50 | 55.10 | 55.36 | 48.84 | 50.13 | 50.50 |
| | Ash | 6.70 | 6.45 | 6.37 | 9.16 | 8.17 | 7.90 |
| Ultimate | Sulphur | 1.60 | 1.32 | 0.90 | 4.24 | 3.78 | 3.50 |
| | Hydrogen | | | 5.26 | | | 5.09 |
| | Carbon | | | 78.31 | | | 74.44 |
| | Nitrogen | | | 1.55 | | | 1.37 |
| | Oxygen | | | 7.61 | | | 7.70 |
| | Calorific value determined: | | | | | | |
| | Calories | 7,845 | | 7,869 | 7,549 | | 7,700 |
| | British thermal units | 14,121 | | 14,164 | 13,588 | | 13,860 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 7,832 | | | 7,517 |
| | British thermal units | | | 14,098 | | | 13,531 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 1.40 | 1.35 | 1.75 | 1.98 | 1.87 | 1.95 |
| | Volatile matter | 36.65 | 37.35 | 36.77 | 40.54 | 40.30 | 39.94 |
| | Fixed carbon | 55.28 | 54.88 | 55.14 | 48.40 | 49.73 | 50.25 |
| | Ash | 6.67 | 6.42 | 6.34 | 9.08 | 8.10 | 7.86 |
| Ultimate | Sulphur | 1.59 | 1.31 | 0.90 | 4.20 | 3.75 | 3.48 |
| | Hydrogen | | | 5.28 | | | 5.18 |
| | Carbon | | | 78.00 | | | 74.07 |
| | Nitrogen | | | 1.54 | | | 1.36 |
| | Oxygen | | | 7.94 | | | 8.10 |
| | Calorific value determined: | | | | | | |
| | Calories | 7,813 | | 7,837 | 7,481 | | 7,661 |
| | British thermal units | 14,063 | | 14,107 | 13,466 | | 13,790 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 7,800 | | | 7,480 |
| | British thermal units | | | 14,040 | | | 13,464 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | West Virginia No. 3. | | | West Virginia No. 4. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| Laboratory sample No. | | 1108 | 1109 | 1252 | 1116 | 1117 | 1262 |
| Loss of moisture on air drying | | 1.50 | 0.80 | 1.30 | 1.10 | 1.20 | 0.50 |
| *Analysis of air-dryed sample.* | | | | | | | |
| Proximate | Moisture | 1.42 | 1.42 | 1.00 | 1.17 | 1.07 | 0.98 |
| | Volatile matter | 29.08 | 29.68 | 30.25 | 29.03 | 28.88 | 28.72 |
| | Fixed carbon | 61.19 | 60.51 | 58.38 | 61.97 | 61.37 | 61.87 |
| | Ash | 8.31 | 8.39 | 10.37 | 7.83 | 8.68 | 8.43 |
| Ultimate | Sulphur | 0.77 | 0.81 | 1.07 | 0.86 | 1.28 | 0.90 |
| | Hydrogen | .......... | .......... | 4.91 | .......... | .......... | 4.85 |
| | Carbon | .......... | .......... | 76.12 | .......... | .......... | 78.21 |
| | Nitrogen | .......... | .......... | 1.44 | .......... | .......... | 1.50 |
| | Oxygen | .......... | .......... | 6.09 | .......... | .......... | 6.11 |
| Calorific value determined: | | | | | | | |
| | Calories | 7,863 | .......... | 7,631 | 7,863 | .......... | 7,856 |
| | British thermal units | 14,153 | .......... | 13,736 | 14,153 | .......... | 14,139 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 7,605 | .......... | .......... | 7,750 |
| | British thermal units | .......... | .......... | 13,689 | .......... | .......... | 13,950 |
| *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 2.90 | 2.21 | 2.29 | 2.26 | 2.26 | 1.48 |
| | Volatile matter | 28.64 | 29.44 | 29.86 | 28.71 | 28.53 | 28.58 |
| | Fixed carbon | 60.27 | 60.03 | 57.62 | 61.29 | 60.63 | 61.55 |
| | Ash | 8.19 | 8.32 | 10.23 | 7.74 | 8.58 | 8.39 |
| Ultimate | Sulphur | 0.75 | 0.80 | 1.06 | 0.85 | 1.26 | 0.90 |
| | Hydrogen | .......... | .......... | 4.99 | .......... | .......... | 4.89 |
| | Carbon | .......... | .......... | 75.13 | .......... | .......... | 77.82 |
| | Nitrogen | .......... | .......... | 1.42 | .......... | .......... | 1.48 |
| | Oxygen | .......... | .......... | 7.17 | .......... | .......... | 6.52 |
| Caloric value determined: | | | | | | | |
| | Calories | 7,745 | .......... | 7,532 | 7,777 | .......... | 7,816 |
| | British thermal units | 13,941 | .......... | 13,558 | 13,999 | .......... | 14,069 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | .......... | .......... | 7,506 | .......... | .......... | 7,711 |
| | British thermal units | .......... | .......... | 13,511 | .......... | .......... | 13,880 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | West Virginia No. 5. | | | West Virginia No. 6. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| Laboratory sample No | | 1144 | 1147 | 1297 | 1175 | 1176 | 1890 |
| Loss of moisture on air drying | | 1.80 | 2.30 | 0.80 | 1.50 | 1.60 | 0.90 |
| *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | 1.04 | 0.80 | 0.65 | 0.80 | 0.51 | 0.64 |
| | Volatile matter | 30.16 | 29.45 | 29.20 | 23.00 | 23.04 | 21.74 |
| | Fixed carbon | 58.16 | 59.24 | 59.97 | 70.23 | 72.85 | 72.53 |
| | Ash | 10.64 | 10.51 | 10.18 | 5.97 | 3.60 | 5.09 |
| Ultimate | Sulphur | 1.02 | 1.15 | 0.99 | 0.74 | 0.76 | 0.66 |
| | Hydrogen | ........ | ........ | 4.78 | ........ | ........ | 4.70 |
| | Carbon | ........ | ........ | 76.36 | ........ | ........ | 83.62 |
| | Nitrogen | ........ | ........ | 1.48 | ........ | ........ | 1.70 |
| | Oxygen | ........ | ........ | 6.21 | ........ | ........ | 4.23 |
| Calorific value determined: | | | | | | | |
| | Calories | 7,623 | ........ | 7,682 | ........ | 8,413 | 8,301 |
| | British thermal units | 13,721 | ........ | 13,828 | ........ | 15,143 | 14,942 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | ........ | ........ | 7,572 | ........ | ........ | 8,208 |
| | British thermal units | ........ | ........ | 13,630 | ........ | ........ | 14,774 |
| *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 2.82 | 3.08 | 1.45 | 2.29 | 2.10 | 1.53 |
| | Volatile matter | 29.62 | 28.77 | 28.97 | 22.65 | 22.67 | 21.54 |
| | Fixed carbon | 57.11 | 57.88 | 59.48 | 69.18 | 71.68 | 71.88 |
| | Ash | 10.45 | 10.27 | 10.10 | 5.88 | 3.55 | 5.05 |
| Ultimate | Sulphur | 1.00 | 1.13 | 0.98 | 0.73 | 0.75 | 0.65 |
| | Hydrogen | ........ | ........ | 4.83 | ........ | ........ | 4.76 |
| | Carbon | ........ | ........ | 75.75 | ........ | ........ | 82.87 |
| | Nitrogen | ........ | ........ | 1.47 | ........ | ........ | 1.68 |
| | Oxygen | ........ | ........ | 6.87 | ........ | ........ | 4.99 |
| Calorific value determined: | | | | | | | |
| | Calories | 7,486 | ........ | 7,621 | ........ | 8,278 | 8,226 |
| | British thermal units | 13,475 | ........ | 13,718 | ........ | 14,900 | 14,807 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | ........ | ........ | 7,511 | ........ | ........ | 8,134 |
| | British thermal units | ........ | ........ | 13,520 | ........ | ........ | 14,641 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | West Virginia No. 7. | | | West Virginia No. 8. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| | Laboratory sample No | 1197 | 1198 | 1595 | 1257 | 1258 | 1515 |
| | Loss of moisture on air drying | 1.70 | 1.50 | 3.20 | 0.60 | 0.60 | 2.60 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 0.80 | 0.63 | 0.76 | 1.31 | 1.25 | 1.60 |
| | Volatile matter | 21.65 | 22.07 | 20.54 | 33.54 | 33.20 | 32.12 |
| | Fixed carbon | 73.33 | 73.70 | 73.61 | 60.25 | 58.75 | 58.92 |
| | Ash | 4.22 | 3.60 | 5.09 | 4.90 | 6.80 | 7.36 |
| Ultimate | Sulphur | 1.10 | 0.91 | 1.20 | 0.64 | 0.89 | 0.92 |
| | Hydrogen | | | 4.38 | | | 5.16 |
| | Carbon | | | 82.41 | | | 78.75 |
| | Nitrogen | | | 1.05 | | | 1.38 |
| | Oxygen | | | 5.87 | | | 6.43 |
| | Calorific value determined: | | | | | | |
| | Calories | | 8,412 | 8,254 | 8,077 | | 7,863 |
| | British thermal units | | 15,142 | 14,851 | 14,539 | | 14,153 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 7,942 | | | 7,884 |
| | British thermal units | | | 14,296 | | | 14,091 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 2.48 | 2.12 | 3.94 | 1.90 | 1.84 | 4.16 |
| | Volatile matter | 21.28 | 21.74 | 19.88 | 33.34 | 33.00 | 31.28 |
| | Fixed carbon | 72.09 | 72.59 | 71.25 | 59.89 | 58.40 | 57.39 |
| | Ash | 4.15 | 3.55 | 4.93 | 4.87 | 6.76 | 7.17 |
| Ultimate | Sulphur | 1.08 | 0.90 | 1.16 | 0.64 | 0.89 | 0.90 |
| | Hydrogen | | | 4.60 | | | 5.32 |
| | Carbon | | | 79.78 | | | 76.70 |
| | Nitrogen | | | 1.01 | | | 1.34 |
| | Oxygen | | | 8.52 | | | 8.57 |
| | Calorific value determined: | | | | | | |
| | Calories | | 8,286 | 7,990 | 8,029 | | 7,659 |
| | British thermal units | | 14,915 | 14,382 | 14,452 | | 13,786 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 7,688 | | | 7,679 |
| | British thermal units | | | 13,838 | | | 13,822 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | West Virginia No. 9. | | | West Virginia No. 10. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Laboratory car sample. |
| Laboratory sample No | | 1208 | 1209 | 1561 | 1240 | 1244 | 1471 |
| Loss of moisture on air drying | | 1.00 | 0.60 | 3.10 | 2.40 | 2.30 | 1.10 |
| *Analysis of air-dried sample.* | | | | | | | |
| Proximate | Moisture | 0.99 | 1.18 | 1.01 | 0.55 | 0.34 | 0.65 |
| | Volatile matter | 34.76 | 32.72 | 29.53 | 18.55 | 19.96 | 18.80 |
| | Fixed carbon | 60.45 | 63.14 | 62.67 | 77.20 | 76.50 | 75.92 |
| | Ash | 3.80 | 2.96 | 6.79 | 3.71 | 3.20 | 4.63 |
| Ultimate | Sulphur | 0.86 | 0.75 | 0.80 | 0.49 | 0.58 | 0.57 |
| | Hydrogen | | | 5.04 | | | 4.58 |
| | Carbon | | | 79.35 | | | 85.91 |
| | Nitrogen | | | 1.63 | | | 1.07 |
| | Oxygen | | | 6.39 | | | 3.24 |
| Calorific value determined: | | | | | | | |
| | Calories | 8,271 | | 7,984 | 8,495 | | 8,439 |
| | British thermal units | 14,888 | | 14,371 | 15,291 | | 15,190 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | | | 7,890 | | | 8,891 |
| | British thermal units | | | 14,202 | | | 15,104 |
| *Analysis corrected to sample as received.* | | | | | | | |
| Proximate | Moisture | 1.98 | 1.77 | 4.08 | 2.93 | 2.63 | 1.75 |
| | Volatile matter | 34.41 | 32.53 | 28.61 | 18.10 | 19.50 | 18.59 |
| | Fixed carbon | 59.85 | 62.76 | 60.73 | 75.85 | 74.74 | 75.08 |
| | Ash | 3.76 | 2.94 | 6.58 | 3.62 | 3.18 | 4.58 |
| Ultimate | Sulphur | 0.85 | 0.74 | 0.77 | 0.48 | 0.57 | 0.56 |
| | Hydrogen | | | 5.23 | | | 4.65 |
| | Carbon | | | 76.89 | | | 84.97 |
| | Nitrogen | | | 1.58 | | | 1.06 |
| | Oxygen | | | 8.95 | | | 4.18 |
| Calorific value determined: | | | | | | | |
| | Calories | 8,188 | | 7,736 | 8,291 | | 8,346 |
| | British thermal units | 14,738 | | 13,925 | 14,924 | | 15,023 |
| Calorific value calculated from ultimate analysis: | | | | | | | |
| | Calories | | | 7,645 | | | 8,298 |
| | British thermal units | | | 13,761 | | | 14,936 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | West Virginia No. 11. | | | West Virginia No. 12. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample A. | Mine sample B. | Laboratory car sample. | Mine sample A. | Mine sample B. | Laboratory car sample. |
| | Laboratory sample No | 1234 | 1235 | 1472 | 1238 | 1242 | 1364 |
| | Loss of moisture on air-drying | 1.50 | 2.60 | 3.30 | 1.60 | 2.90 | 1.10 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 0.72 | 0.46 | 0.80 | 0.32 | 0.60 | 0.62 |
| | Volatile matter | 18.56 | 18.74 | 16.90 | 19.68 | 19.45 | 18.05 |
| | Fixed carbon | 75.40 | 76.11 | 70.80 | 75.54 | 75.93 | 74.38 |
| | Ash | 5.32 | 4.69 | 11.50 | 4.46 | 4.02 | 6.95 |
| Ultimate | Sulphur | 0.45 | 0.51 | 0.53 | 0.53 | 0.75 | 0.69 |
| | Hyrogen | .......... | .......... | 4.03 | .......... | .......... | 4.36 |
| | Carbon | .......... | .......... | 79.12 | .......... | .......... | 83.63 |
| | Nitrogen | .......... | .......... | 1.04 | .......... | .......... | 1.34 |
| | Oxygen | .......... | .......... | 3.78 | .......... | .......... | 3.08 |
| | Calorific value determined: | | | | | | |
| | Calories | 8,343 | .......... | 7,761 | 8,427 | 8,428 | 8,185 |
| | British thermal units | 15,017 | .......... | 13,970 | 15,169 | 15,170 | 14,733 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | .......... | .......... | 7,631 | .......... | .......... | 8,144 |
| | British thermal units | .......... | .......... | 13,736 | .......... | .......... | 14,659 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 2.21 | 3.05 | 4.07 | 1.92 | 3.48 | 1.72 |
| | Volatile matter | 18.28 | 18.26 | 16.34 | 19.36 | 18.89 | 17.85 |
| | Fixed carbon | 74.26 | 74.12 | 68.47 | 74.33 | 73.73 | 73.56 |
| | Ash | 5.25 | 4.57 | 11.12 | 4.89 | 3.90 | 6.87 |
| Ultimate | Sulphur | 0.44 | 0.50 | 0.51 | 0.52 | 0.73 | 0.68 |
| | Hydrogen | .......... | .......... | 4.27 | .......... | .......... | 4.43 |
| | Carbon | .......... | .......... | 76.51 | .......... | .......... | 82.71 |
| | Nitrogen | .......... | .......... | 1.00 | .......... | .......... | 1.33 |
| | Oxygen | .......... | .......... | 6.59 | .......... | .......... | 3.98 |
| | Calorific value determined: | | | | | | |
| | Calories | 8,218 | .......... | 7,505 | 8,292 | 8,184 | 8,095 |
| | British thermal units | 14,792 | .......... | 13,509 | 14,926 | 14,731 | 14,571 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | .......... | .......... | 7,379 | .......... | .......... | 8,054 |
| | British thermal units | .......... | .......... | 13,282 | .......... | .......... | 14,497 |

*Analyses of mine and laboratory car samples of coal*—Continued.

| | | Wyoming No. 1. | | | Wyoming No. 2. | | |
|---|---|---|---|---|---|---|---|
| | | Mine sample B. | Mine sample A. | Laboratory car sample. | Mine sample B. | Mine sample A. | Gas-producer sample. |
| | Laboratory sample No | 1368 | 1369 | 1479 | 1376 | 1377 | 1571 |
| | Loss of moisture on air drying | 5.00 | 4.50 | 6.00 | 4.70 | 3.80 | 6.90 |
| | *Analysis of air-dried sample.* | | | | | | |
| Proximate | Moisture | 17.89 | 17.74 | 17.69 | 4.09 | 5.64 | 2.73 |
| | Volatile matter | 37.81 | 38.91 | 37.96 | 38.96 | 37.51 | 37.61 |
| | Fixed carbon | 40.75 | 38.21 | 39.56 | 33.97 | 35.06 | 37.40 |
| Ultimate | Ash | 3.55 | 5.14 | 4.79 | 22.98 | 21.80 | 22.26 |
| | Sulphur | 0.63 | 0.55 | 0.63 | 5.18 | 4.50 | 4.17 |
| | Hydrogen | | | 5.06 | | | 4.54 |
| | Carbon | | | 58.41 | | | 55.29 |
| | Nitrogen | | | 1.09 | | | 0.80 |
| | Oxygen | | | 28.99 | | | 12.94 |
| | Calorific value determined: | | | | | | |
| | Calories | 5,728 | | 5,753 | 5,660 | | 5,758 |
| | British thermal units | 10,340 | | 10,355 | 10,188 | | 10,364 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 5,582 | | | 5,567 |
| | British thermal units | | | 10,048 | | | 10,021 |
| | *Analysis corrected to sample as received.* | | | | | | |
| Proximate | Moisture | 22.00 | 21.44 | 22.68 | 8.60 | 9.23 | 9.44 |
| | Volatile matter | 35.92 | 37.16 | 35.68 | 37.15 | 36.08 | 35.02 |
| | Fixed carbon | 38.71 | 36.49 | 37.19 | 32.37 | 33.72 | 34.82 |
| Ultimate | Ash | 3.37 | 4.91 | 4.50 | 21.90 | 20.97 | 20.72 |
| | Sulphur | 0.60 | 0.53 | 0.59 | 4.94 | 4.33 | 3.91 |
| | Hydrogen | | | 6.39 | | | 5.00 |
| | Carbon | | | 54.91 | | | 51.46 |
| | Nitrogen | | | 1.02 | | | 0.74 |
| | Oxygen | | | 32.59 | | | 18.17 |
| | Calorific value determined: | | | | | | |
| | Calories | 5,442 | | 5,408 | 5,394 | | 5,361 |
| | British thermal units | 9,796 | | 9,734 | 9,709 | | 9,650 |
| | Calorific value calculated from ultimate analysis: | | | | | | |
| | Calories | | | 5,247 | | | 5,183 |
| | British thermal units | | | 9,445 | | | 9,329 |

# WASHING TESTS.

By John D. Wick.

## INTRODUCTION.

The washery plant was constructed after designs furnished by Roberts & Shaefer Company, of Chicago. It was under the immediate supervision of the writer, who was assisted by Mr. Edward Moore. The chemical analyses given in the report were made at the laboratory of the testing plant under the direction of Prof. N. W. Lord. The steam tests mentioned were made in the boiler room under the direction of Prof. L. C. Breckenridge.

Owing to the small appropriation and the great cost of all material and labor on the Exposition grounds, the washery plant was made as small as possible, consisting of four 35-ton storage bins and two bins of half the size for holding material to be washed. On account of the small size of the bins holding material to be washed, it was impossible to wash an entire carload of coal in one operation. Washing tests, therefore, had to be made in lots of 10 tons or less, and for that reason the best results were not always obtained and not many such tests were made. The lack of adequate storage facilities and the constant demand upon the conveying apparatus for supplying fuel to the boiler room, the gas-producer plant, the coke ovens, and the briquetting machines made it frequently impossible to weigh carefully the washed product and the refuse material. Although complete washing tests were made in only a few instances, considerable coal was washed for coking, but in small amounts, only sufficient for charging a single oven.

As noted in the general description of the plant (pages 10–13), the washing apparatus consisted of a New Century jig and a modified Stewart jig. The former is desiged for washing very fine material, whereas the latter will wash coal composed of pieces crushed to a diameter of 1½ inches. The tests made were not sufficiently complete to determine the efficiency of these jigs, but considerable difficulty was experienced with the New Century jig, the material being so fine that it packed in the bottom of the jig and thus prevented the discharge of the refuse material. The modified Stewart jig gave very good satisfaction, and seems very well adapted to most kinds of coal.

## SPECIAL TESTS.

Complete washing tests were made upon the samples named below, and with the results stated.

### ILLINOIS NO. 2.

This sample consisted of a carload of slack from the O'Fallon mine of the Western Anthracite Coal and Coke Company, near O'Fallon, Ill. The coal in this district produces considerable slack, which finds a fairly ready market in St. Louis, but the price received is hardly sufficient to pay for transportation. Washing has been introduced at a number of plants to improve the quality of the slack, with very promising results, and this test was made with the object of showing what improvement could be effected in this particular part of the district. The slack washed consisted of 27,280 pounds, which was carefully sampled before and after washing.

This coal was washed in the modified Stewart jig. The weight of the washed coal was not ascertained, but the refuse weighed 3,520 pounds.

The first column of the appended table shows the composition of the raw coal as far as its content of moisture, ash, and sulphur is concerned. The second column gives the same data for the washed coal and the third column for the refuse.

*Analyses of Illinois No. 2 coal and refuse, showing effect of washing.*

| | Raw coal. | Washed coal. | Refuse. |
|---|---|---|---|
| Moisture | 12.03 | 19.07 | 15.99 |
| Ash | 22.44 | 9.42 | 53.08 |
| Sulphur | 4.00 | 3.35 | 9.06 |
| Weight, in pounds | 27,280 | .......... | 3,520 |

A comparison of these figures shows that the percentage of ash was reduced from 22.44 to about 9.42 by washing, but the greater amount of moisture in the washed coal sample makes direct comparison impossible. If the analysis of the washed coal had been determined on the same basis as that of the raw coal, the percentage of ash in the latter would be about 10 per cent. This shows a reduction of fully 12 per cent in ash as a result of washing a small lot through the Stewart jig, but washing on a commercial scale might improve this result considerably.

The sulphur shows a slight reduction, about one-half of 1 per cent. It would seem from these figures either that the major portion of the sulphur is so intimately combined with the coal as to be inseparable from it, or that the sulphur is present in thin flakes which pass over *with the coal.*

The next important item in successful washing is the determination of the amount of coal retained in the refuse or, in other words, wasted. From the composition of the refuse and its weight the amount of coal in the refuse is about 1,000 pounds, or about 3½ per cent of the original charge, or about 5½ per cent of the coal in the original charge.

The effect of washing in improving the quality of this coal is shown by steam tests Nos. 18 and 19. Test No. 18 was made on lump coal from this mine, and test No. 19 was made on the washed slack from the same mine.

The following brief results of these tests are given:

*Steam tests of lump and washed coal (Illinois No. 2).*

| Coal tested. | Total coal consumed. | Horse-power developed by boiler. | Dry coal burned per square foot of grate surface, per hour. | Equivalent evaporation from and at 212° F. per pound of dry coal. | Dry coal per indicated horse-power hour. | Dry coal per electrical horse-power hour. |
|---|---|---|---|---|---|---|
| | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* |
| Lump coal | 11,124 | 211.4 | 24.90 | 7.21 | 3.92 | 4.85 |
| Washed slack | 10,096 | 210.2 | 22.36 | 8.00 | 3.54 | 4.38 |

A small amount of this coal was also washed for coking purposes, and the improvement may be noted by comparing the results obtained from this charge with those obtained from a charge of Illinois No. 1, which consists of lump coal from the same mine. This test was made on a charge of 9,000 pounds of unwashed coal, which was burned for 43 hours. It produced some small pieces of coke mixed with charred coal and ash. This coal contained 15.95 per cent of ash, and probably for this reason failed to coke.

The second test was made on Illinois No. 2 washed. The amount of coal charged was 9,000 pounds, which, after burning 64 hours, yielded 3,389 pounds of mixed hard and soft coke and 352 pounds of breeze and ash.

From this report it is obvious that the raw lump coal did not coke, but the washed slack produced coke of fair quality, although the percentage of the yield is so small that it probably would not be commercially successful.

The improvement by washing is also shown by the chemical analyses of the coal before and after washing, as follows:

*Analyses showing effect of washing Illinois No. 2 coal.*

| | Raw coal. | Washed coal for coking. |
|---|---|---|
| Ash | 22.44 | 9.19 |
| Sulphur | 4.00 | 3.03 |

### INDIANA NO. 1.

This was a sample of run-of-mine coal from mine of the J. Woolley Coal Company, located at Mildred, Sullivan County, Ind. No washing is now done in this district, but the coal contains considerable impurity, and the operators are very desirous of obtaining information concerning the effectiveness of washing operations.

The charge, consisting of 15,250 pounds of run-of-mine coal, was passed through rolls having an aperture of 1¼ inches, and then washed through the modified Stewart jig.

After the coal was crushed it was sampled and analyzed, giving the figures shown in the column marked "Raw coal" in the accompanying table. After washing it was again sampled and analyzed, and the results are given in the second column of the table. The refuse was also sampled and analyzed, with the results shown in the third column.

*Analyses of Indiana No. 1 coal and refuse, showing effect of washing.*

| | Raw coal. | Washed coal. | Refuse. |
|---|---|---|---|
| Moisture | 11.40 | 16.72 | 14.85 |
| Ash | 13.40 | 7.16 | 31.71 |
| Sulphur | 2.50 | 2.23 | 5.68 |
| Weight, in pounds | 15,250 | 12,620 | 3,485 |
| Coal, pounds | 11,468 | 9,600 | 1,860 |

The results of this test are not so satisfactory as that of Illinois No. 2, but it is probable that equally good results could be obtained by further trials.

The test of Indiana No. 1 is satisfactory as far as the reduction of the impurities is concerned, but it is not satisfactory when it is considered that about 1,860 pounds of coal (exclusive of moisture and ash) passed over with the refuse matter. This is probably too great a loss for commercial success, but, as noted above, there is every reason to believe that washing could be done that would result in a much cleaner coal and at the same time lose but little coal with the refuse material.

A steam test was made of this coal after washing, but no test was made on the raw coal, so it is impossible to say how great the improvement was except from the chemical analyses. The steam test on the washed coal is given in test No. 68, page 81.

A washing test was also made on about 4½ tons of this coal for coking purposes. No test was made in the coke ovens of the raw coal, hence the coke produced can not be compared, but the chemical analyses show the improvement effected, as follows:

*Analyses of Indiana No. 1 coal, showing effect of washing.*

| | Raw coal. | Washed coal for coking. |
|---|---|---|
| Ash | 13.40 | 7.04 |
| Sulphur | 2.50 | 2.03 |

### MISSOURI NO. 3.

This sample consisted of a carload of slack from the Mendota Coal and Mining Company, operating at Mendota, Putnam County, Mo. Like most of the coals from northern Missouri and southern Iowa, this coal contains a heavy percentage of impurities, and the operators are very desirous of improving their product. It was for this reason that the above test was undertaken, as the results of this test will doubtless apply to the entire field in Putnam County, Mo., and Appanoose County, Iowa.

The sample, consisting of 14,000 pounds of slack, was passed through rolls having an aperture of 1¼ inches and then washed through the modified Stewart jig. After the coal was crushed it was sampled and analyzed, giving the figures shown in the column marked "Raw coal" in the accompanying table. After washing it was again sampled and analyzed, with the results given in the second column of the table. The refuse was also sampled and analyzed, with the results shown in the third column.

*Analyses of Missouri No. 3 coal and refuse, showing effect of washing.*

| | Raw coal. | Washed coal. | Refuse. |
|---|---|---|---|
| Moisture | 14.37 | 23.90 | 23.38 |
| Ash | 28.39 | 7.59 | 36.70 |
| Sulphur | 4.30 | 2.89 | 3.94 |
| Weight, in pounds | 14,000 | .......... | 4,931 |
| Calculated weight of coal, in pounds | 8,013 | 6,045 | 1,968 |

The refuse weighed 4,931 pounds. It appears from the chemical analysis that the raw coal contained 8,013 pounds of coal exclusive of moisture and ash, and the refuse 1,968 pounds. The washed coal, therefore, must have been the difference, or 6,045 pounds.

The results given above show great improvement. The slack was extremely dirty, as the analysis shows. Washing reduced the ash from 28.39 per cent in the raw coal to 7.59 per cent in the washed coal. The sulphur was likewise reduced from 4.30 to 2.89 per cent. On the whole the reduction in the impurities is highly satisfactory, but the amount of coal lost in the operation (1,968 pounds) is too great for economical work. It is possible that better results might have been obtained if the tests had been continued, but enough has been done to show that great improvement may be made in the quality of the slack coal in this field by washing.

The improvement in washing this slack was still further tested by steam tests made on the raw coal and also on the washed product.

*Steam tests showing effect of washing Missouri No. 3 coal.*

| Coal tested. | Total coal consumed. | Horsepower developed by boiler. | Dry coal burned per sq. ft. of grate surface, per hour. | Equivalent evaporation from and at 212° F., per pound of dry coal. | Dry coal, per indicated horsepower hour. | Dry coal, per electrical horsepower hour. |
|---|---|---|---|---|---|---|
| | *Pounds.* | | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* |
| Raw slack | 10,828 | 149.5 | 21.85 | 5.82 | 4.86 | 6.00 |
| Washed slack | 11,044 | 189.7 | 21.72 | 7.43 | 3.81 | 4.70 |

A washing test was also made of about 6½ tons of this coal for coking purposes. No trial was made of the raw coal in the oven, so it is impossible to make a direct comparison of the coke, but the chemical analyses give some idea of the improvement produced.

*Analyses showing effect of washing Missouri No. 3 coal.*

| | Raw coal. | Washed coal for coking. |
|---|---|---|
| Ash | 28.39 | 7.24 |
| Sulphur | 4.30 | 2.74 |

## GENERAL TESTS.

In addition to the above-mentioned special tests, a great many charges of coal for the coke ovens were washed, and generally with fairly satisfactory results, considering the small amount of coal in each charge.

As previously stated, the congested condition of the conveying apparatus prevented the weighing of any of these small charges after washing, but some idea of the effects of washing may be gained by comparing the chemical analyses made before and after washing and by noting the character of coke produced when tests were made on washed and unwashed material.

*Alabama No. 1.*—Lump coal from mine No. 8, Ivy Coal and Iron Company, Horse Creek, Ala.

Coking tests were made of both washed and unwashed coal from this mine. The first charge of unwashed coal consisted of 8,000 pounds. Although the coking process was continued for 88 hours, the coke was very light and spongy and high in ash. Washing reduced the percentage of impurities but slightly, though it is believed that with more thorough washing a coke of good quality might be obtained.

*Analyses showing effect of washing Alabama No. 1 coal.*

| | Raw coal. | Washed coal for coking. |
|---|---|---|
| Ash | 13.88 | 11.06 |
| Sulphur | .76 | .89 |

*Arkansas No. 6.*—Slack coal from mine No. 18, Western Coal and Mining Company, Jenny Lind, Ark.

About 5 tons of this coal were washed for coking purposes. No coke was produced, but the chemical analyses show improvement.

*Analyses showing effect of washing Askansas No. 6 coal.*

| | Raw coal. | Washed coal for coking. |
|---|---|---|
| Ash | 13.81 | 6.22 |
| Sulphur | 1.26 | 1.22 |

*Illinois No. 3*—Run-of-mine coal from mine No. 3, Southern Illinois Coal Mining and Washing Company, Marion, Ill.

Two coking tests were made of this coal, one of raw coal and the other washed for the purpose. These tests permit direct comparison, and the effect of washing is plainly apparent. The unwashed charge of 9,000 pounds was tested for 43 hours. The coal lay dead in the oven, burning on top, but did not coke. In the second test the charge (washed) consisted of 13,000 pounds and was burned 90 hours, yielding 6,378 pounds of coke, which was very brittle and which broke up in handling into fine-fingered pieces.

The chemical composition of the two grades of coal is shown below:

*Analyses showing effect of washing Illinois No. 3 coal.*

| | Raw coal for coking. | Washed coal for coking. |
|---|---|---|
| Ash | 10.59 | 5.86 |
| Sulphur | 1.45 | 1.41 |

*Illinois No. 5.*—No. 5 washed slack from mine No. 1, Donk Brothers Coal and Coke Company, Collinsville, Ill.

The slack at this plant is washed and divided into five grades, and the sample consisted of No. 5, or the finest grade produced. The particles of coal were all less than one-fourth inch in diameter. The coal was washed at the mine, but owing to the overcrowded condition of the washery this grade of coal carried a considerable amount of impurities, mainly as a clay coating on the particles of coal. As the coal did not coke in the condition received, a charge consisting of about 6 tons was rewashed in the New Century jig. The coal was extremely wet when charged, chilling the oven, and no coke was produced. Although the coking test was not successful, the rewashing showed a distinct gain in the quality of the coal, but considerable coal was lost in the operation. Chemical analyses show the following results:

*Analyses showing effect of washing Illinois No. 5 coal.*

| | First charge washed. | Second charge rewashed. | Third charge washed. |
|---|---|---|---|
| Ash | 17.56 | 9.18 | 18.16 |
| Sulphur | 3.25 | 2.71 | 3.44 |

*Indian Territory No 2.*—Run-of-mine coal from mine No. 8, Rock Island Coal Company, Hartshorne, Ind. T.

Tests were made on this coal in the raw state and also after washing. The unwashed charge (9,000 pounds) was burned for 66 hours, and yielded 5,725 pounds of coke and 580 pounds of breeze and ash. The coke was very soft, shattered, and brittle. The washed charge was burned for 65 hours, producing a coke which had a fairly good ring. It showed considerable improvement in appearance over the coke made from unwashed coal. The change in the impurities is shown by the following analyses.

*Analyses showing effect of washing Indian Territory No. 2 coal.*

| | Coal. | | Coke. | |
|---|---|---|---|---|
| | Raw. | Washed. | From raw coal. | From washed coal. |
| Ash | 9.99 | 6.33 | 14.43 | 11.12 |
| Sulphur | 1.47 | 1.43 | 1.50 | 1.75 |

*Indian Territory No 3.*—Run-of-mine coal from mine No. 1, D. Edwards & Sons, Edwards, Ind. T.

Coking tests were made on this coal in both the raw and the washed condition, but the washing does not appear to have been successful in reducing materially either the sulphur or the ash. No coke was made from either charge. The amount of impurities is shown in the following table:

*Analyses showing effect of washing Indian Territory No. 3 coal.*

| | Raw coal for coking. | Washed coal for coking. |
|---|---|---|
| Ash | 9.75 | 7.49 |
| Sulphur | 3.16 | 3.20 |

*Indian Territory No 5.*—Slack and pea coal from mine No. 7, Western Coal and Mining Company, Lehigh, Ind. T.

About 5 tons of this slack were washed for a coking test, but without satisfactory results so far as the production of coke is concerned. The improvement effected by washing is shown in the following analyses:

*Analyses showing effect of washing Indian Territory No. 5 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 25.05 | 8.14 |
| Sulphur | 3.95 | 2.90 |

*Iowa No. 1.*—Lump and fine coal from mine No. 2, Anchor Coal Company, Laddsdale, Iowa.

About 5 tons of this coal were washed for a coking test, but the coal was not tried in a raw condition, and consequently the coking test affords no clue to the improvement made by washing. The change is shown by the chemical analyses.

*Analyses showing effect of washing Iowa No. 1 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 16.0 | 10.25 |
| Sulphur | 5.03 | 4.61 |

*Iowa No. 2.*—Run-of-mine coal from mine No. 6, Mammoth Vein Coal Company, Hamilton, Iowa.

About 5½ tons of coal were washed for a coking test. The reduction of impurities effected by washing was not great, as shown by the following analyses:

*Analyses showing effect of washing Iowa No. 2 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 15.22 | 10.28 |
| Sulphur | 4.66 | 3.93 |

*Iowa No. 3.*—Lump coal from mine No. 4, Gibson Coal Mining Company, Altoona, Iowa.

About 4½ tons of this coal were washed for a coking test. The improvement in the quality of the coal effected by washing is shown in the following analyses:

*Analyses showing effect of washing Iowa No. 3 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 14.01 | 8.03 |
| Sulphur | 6.15 | 4.55 |

*Iowa No. 4.*—Lump coal from mine No. 3, Centerville Block Coal Company, Centerville, Iowa.

A charge consisting of about 4½ tons of this coal was washed for coking purposes. The results were not so satisfactory as those obtained on other samples from this State. The analyses are given below:

*Analyses showing effect of washing Iowa No. 4 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 10.96 | 7.14 |
| Sulphur | 4.26 | 3.59 |

*Iowa No. 5.*—Run-of-mine coal from mine No. 1, Inland Fuel Company, Chariton, Iowa.

A charge consisting of nearly 5 tons of this coal was washed for a coking test, but the coal did not coke, although the washing was fairly successful in reducing the impurities, as shown by the following analyses:

*Analyses showing effect of washing Iowa No. 5 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 12.63 | 7.93 |
| Sulphur | 3.19 | 2.28 |

*Kentucky No. 3.*—Run-of-mine coal from Barnsley mine, St. Bernard Mining Company, Earlington, Ky.

A charge of about 5½ tons of this coal was washed for coking purposes. It produced a coke of fair quality, although little reduction was made in the impurities by washing. The analyses are as follows:

*Analyses showing effect of washing Kentucky No. 3 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 10.06 | 7.40 |
| Sulphur | 3.52 | 2.51 |

*Kentucky No. 4.*—Run-of-mine coal from Wheatcroft Coal and Mining Company, Wheatcroft, Ky.

A charge of about 5¼ tons of this coal was washed for a coking test. Washing was very much more successful in this case than in that of the other coal from Kentucky. The change is shown in the following table:

*Analyses showing effect of washing Kentucky No. 4 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 14.18 | 6.05 |
| Sulphur | 4.54 | 2.74 |

*Missouri No. 2.*—Run-of-mine coal from mine No. 8, Northwestern Coal and Mining Company, Bevier, Mo.

A charge of about 6½ tons of this coal was washed for a coking test, with the following result:

*Analyses showing effect of washing Missouri No. 2 coal.*

| | Car sample. | Washed coal for coking. |
|---|---|---|
| Ash | 16.86 | 7.76 |
| Sulphur | 5.16 | 3.24 |

*West Virginia No. 2.*—Run-of-mine coal from Pitcairn mine, Pitcairn Coal Company, Clarksburg, W. Va.

Two coking tests were made on this sample, one of raw coal and one of washed coal. This affords an additional means of judging of the effectiveness of washing, for the coke shows improvement in the second (washed coal) test, which can be accounted for only by better quality of coal. The following table shows the results:

*Analyses showing the effect of washing West Virginia No. 2 coal.*

| | Coal. | | Coke. | |
|---|---|---|---|---|
| | Raw. | Washed. | From raw coal. | From washed coal. |
| Ash | 8.22 | 7.05 | 14.95 | 11.40 |
| Coke | 3.38 | 2.84 | 3.40 | 2.24 |

*West Virginia No. 3.*—Run-of-mine coal from West Virginia Coal Company, Richard, W. Va.

This coal was tested in the coke ovens in both the raw and the washed condition, and a comparison of results shows that washing as done in this test had little effect. The charge that was washed weighed about 7½ tons.

*Analyses showing effect of washing West Virginia No. 3 coal.*

| | Coal. | | Coke. | |
|---|---|---|---|---|
| | Raw. | Washed. | From raw coal. | From washed coal. |
| Ash | 9.75 | 9.01 | 18.18 | 14.27 |
| Sulphur | .99 | 1.18 | .93 | 1.19 |

*West Virginia No. 4.*—Run-of-mine coal from West Virginia Coal Company, Bretz, W. Va.

Two coking tests were made of this coal, one in the raw and one in a washed condition. The charge that was washed weighed about 5 tons. The results of these tests, as shown by the analyses of the coal and the coke, are not in harmony, so it is difficult to determine the effect of washing.

The analyses are as follows:

*Analyses showing effect of washing West Virginia No. 4 coal.*

| | Coal. | | Coke. | |
|---|---|---|---|---|
| | Raw. | Washed. | From raw coal. | From washed coal. |
| Ash | 8.39 | 7.53 | 11.85 | 13.23 |
| Sulphur | .86 | .74 | .82 | .69 |

The change in the amount of impurities is so slight that the irregularities of sampling are probably responsible for the difference in the figures.

*West Virginia No. 5.*—Lump and nut coal from mine of Davis Colliery Company, Coalton, W. Va.

Three coking tests were made of this coal to determine the possibility of reducing the ash to within the limit of coke for blast furnace use. The first charge consisted of about 6½ tons of raw coal; the second charge contained about 7 tons of coal crushed in rolls to 1½ inches in size and then washed; the third charge contained about 5½ tons of coal, which was pulverized in the Williams mill and washed in the New Century jig. The results of washing are shown by the following analyses:

*Analyses showing effect of washing West Virginia No. 5 coal.*

| | Coal. | | | Coke. | | |
|---|---|---|---|---|---|---|
| | Raw. | Washed. | Pulverized and washed. | From raw coal. | From washed coal. | From pulverized and washed coal. |
| Ash | 10.73 | 10.28 | 8.19 | 19.14 | 14.81 | 15.98 |
| Sulphur | .90 | .91 | .79 | .77 | .83 | .82 |

*West Virginia No. 9.*—Run-of-mine coal from Vulcan mine, Mount Carbon Coal Company (Limited), Powellton, W. Va.

This coal was tested in the coke ovens in both the raw and the washed condition. The charge of raw coal contained about 4½ tons and of washed coal about 6 tons. The coke from the washed coal showed much improvement over that from the unwashed coal. The change in composition is shown by the following analyses:

*Analyses showing effect of washing West Virginia No. 9 coal.*

| | Coal. | | Coke. | |
|---|---|---|---|---|
| | Raw. | Washed. | From raw coal. | From washed coal. |
| Ash | 8.07 | 4.51 | 9.15 | 7.38 |
| Sulphur | .83 | .90 | .82 | .77 |

*West Virginia No. 12.*—Run-of-mine coal from mine of the Big Sandy Coal and Coke Company, Big Sandy, W. Va.

Two coking tests were made of this coal, one of raw coal and one of washed coal. The first charge consisted of about 5½ tons of coal and the second charge of about 4½ tons. No particular change was noted in the coke made from the two grades of coal, but the analyses show a reduction of ash as follows:

*Analyses showing effect of washing West Virginia No. 12 coal.*

| | Coal. | | Coke. | |
|---|---|---|---|---|
| | Raw. | Washed. | From raw coal. | From washed coal. |
| Ash | 6.16 | 4.90 | 9.43 | 7.55 |
| Sulphur | .97 | 1.11 | .83 | 1.01 |

# STEAM TESTS.

By L. P. Breckenridge.

## INTRODUCTION.

The following is a preliminary report of the steam tests of the coals burned under the boilers of the United States Geological Survey coal-testing plant at the Louisiana Purchase Exposition, at St. Louis.

This report consists of (*a*) a short description of the methods used in testing and a table giving the important dimensions of the boilers used; (*b*) a list of the various coals tested, and (*c*) some of the important results of the tests.

## METHODS OF CONDUCTING TESTS.

The method of testing fuels under boilers has been a subject of discussion for many years by the members of the American Society of Mechanical Engineers, and as the result of this discussion a standard method of conducting tests and of reporting the results has received the approval of this society. The steam tests of coal at the Geological Survey testing plant have been conducted and the results reported in accordance with these methods and forms.

The number of tests made was 78. The duration of each was planned for ten hours, and was as near this time as the conditions at the close of a test would permit. An experienced and careful fireman hand-fired all of the coals tested.

## DESCRIPTION OF THE BOILERS.

There were two 210-horsepower Heine safety boilers provided for these tests. They were exactly similar in construction and setting. Each was provided with its own stack, 115 feet high and 37 inches in diameter. Each boiler was fed by its own independent injector, and no other means of supplying water to the boilers was provided. The ends of the blow-off pipes were visible during all tests.

In the next table the leading proportions of one of the boilers are given.

The boilers were cleaned externally after each test. The interior condition of each boiler was practically clean during all of the tests.

*Leading proportions of Heine water-tube boilers used in coal-testing plant.*

| Item No. | Item. | Dimensions. |
|---|---|---|
| 1 | Rated capacity of boiler........................horsepower.. | 210 |
| 2 | Water-heating surface..........................square feet.. | 2,031 |
| 3 | Superheating surface........................................ | None. |
| 4 | Grate area..........................................square feet.. | 40.6 |
| 5 | Kind of draft................................................ | Natural. |
| 6 | Height of steel stack....................................feet.. | 115 |
| 7 | Area of steel stack.............................square feet.. | 7.67 |
| 8 | Number of tubes.............................................. | 116 |
| 9 | Outside diameter of tubes..........................inches.. | 3.5 |
| 10 | Steam pressure......................................pounds.. | 85 |
| 11 | Observations taken every 20 minutes. | |
| 12 | Approximate duration of each trial...............hours.. | 10 |

## DESCRIPTION OF THE STEAM ENGINE AND GENERATOR.

The steam generated by the boilers was largely used by an Allis-Corliss engine of 250-horsepower capacity, and the power of the engine was absorbed by a 200-kilowatt direct current, 240-volt Bullock generator, to which the engine was belted.

The engine was the simple noncondensing type. The cylinder was 22 inches in diameter and the stroke was 42 inches. The engine ran about 80 revolutions per minute.

Numerous tests made to determine the steam consumption of the engine, as well as the mechanical efficiency of the engine and generator together, gave the following average results:

*Average results of engine and generator tests.*

| | |
|---|---|
| Number of pounds of steam used per hour per indicated horsepower developed by the engine.................................................. | 23.60 |
| Mechanical efficiency of the engine and generator combined (per cent)...... | 81 |

From these figures has been calculated the electrical horsepower delivered to the switchboard for such tests of the boilers as were not accompanied by a test of the engine and generator.

## COALS TESTED.

On pages 76 to 78 the coals tested are arranged alphabetically according to the States from which they were obtained. Some of these coals were washed and some were briquetted. In all tests of coals other than slack, the coal was first passed through rolls having an opening of 1½ inches, which crushed it to uniform size. On pages 80 to 83 is a table which gives some of the results of the tests, and in which the character of the coal used in each boiler trial is clearly indicated.

## LIST OF COALS TESTED.

Alabama No. 1. First test: Lump and nut coal from mine No. 8 of the Ivy Coal and Iron Company, Horse Creek, Ala.

Second test: Large briquettes containing 7 per cent of pitch binder. See p. 148.

Alabama No. 2. Lump, nut, and pea coal from mine No. 5 of the Galloway Coal Company, Carbon Hill, Ala.

Arkansas No. 1. First test: Lump and nut coal from mine No. 3 of the Central Coal and Coke Company, Huntington, Ark.

Second test: Large briquettes containing 9½ per cent of pitch binder. See p. 148.

Arkansas No. 2. First test: Lump coal from mine No. 12 of the Central Coal and Coke Company, Bonanza, Ark.

Second test: Large briquettes containing 11 per cent of pitch binder. See p. 149.

Arkansas No. 3. First test: Lump and slack coal from mine No. 18 of the Western Coal and Mining Company, Jenny Lind, Ark.

Second test: Large briquettes containing 8.7 per cent of pitch binder. See p. 149.

Arkansas No. 4. First test: Large briquettes made from slack coal from several Arkansas mines, furnished by the Western Coal and Mining Company, St. Louis, Mo. Briquettes contained 6 per cent of pitch binder. See p. 151.

Second test: Small briquettes made with patent binder. See p. 151.

Arkansas No. 5. Lump and slack coal from mine No. 4 of the Western Coal and Mining Company, Coal Hill, Ark.

Colorado No. 1. Run-of-mine black lignite, from Simpson mine, of the Northern Coal and Coke Company, Layfayette, Colo.

Illinois No. 1. Lump and nut coal from mine No. 1 of the Western Anthracite Coal and Coke Company, near O'Fallon, Ill.

Illinois No. 2. Washed slack coal, from same as Illinois No. 1.

Illinois No. 3. Run-of-mine coal from mine No. 3 of the Southern Illinois Coal Mining and Washing Company, near Marion, Ill.

Illinois No. 4. First test: Lump coal from mine No. 3 of the Donk Brothers Coal and Coke Company, Troy, Ill.

Second test: Same as above.

Illinois No. 6. Run-of-mine coal from shaft No. 1 of Clover Leaf Coal Company, Coffeen, Ill.

Indiana No. 1. Washed run-of-mine coal from Mildred mine of the J. Woolley Coal Company, Mildred, Ind.

Second test: Large briquettes made from washed coal, containing 7 per cent of pitch binder. See p. 154.

Indiana No. 2. Run-of-mine coal from Electric mine of the T. D. Scales Coal Company, Boonville, Ind.

Indian Territory No. 1. Lump and slack coal from mine No. 1 of the Whitehead Coal and Mining Company, Henryetta, Ind. T.

Indian Territory No. 2. Run-of-mine coal from mine No. 8 of the Rock Island Coal Company, Hartshorne, Ind. T.

Indian Territory No. 3. Run-of-mine coal from mine No. 1 of D. Edwards & Son, Edwards, Ind. T.

Indian Territory No. 4. Lump coal from mine No. 5 of the Western Coal & Mining Company, Lehigh, Ind. T.

Iowa No. 1. Lump and fine coal from mine No. 2 of the Anchor Coal Company, Laddsdale, Iowa.

Iowa No. 2. Run-of-mine coal from mine No. 6 of the Mammoth Vein Coal Company, near Hamilton, Iowa.

Iowa No. 3. Lump coal from mine No. 4 of the Gibson Coal Mining Company, Altoona, Iowa.

Iowa No. 4. First test: Lump coal from mine No. 3 of the Centerville Block Coal Company, Centerville, Iowa.

Second test: Large briquettes containing 8 per cent of pitch. See p. 158.

Iowa No. 5. Run-of-mine coal from Inland mine No. 1 of the Inland Fuel Company, Chariton, Iowa.

Kansas No. 1. Run-of-mine coal from mine No. 10 of the Western Coal and Mining Company, Fleming, Kans.

Kansas No. 2. First test: Lump and nut coal from mine No. 11 of the Western Coal and Mining Company, Yale, Kans.

Second test: Same as above except washed.

Kansas No. 3. First test: Run-of-mine coal from mine No. 9 of the Southern Coal and Mercantile Company, Scammon, Kans.

Second test: Same as above.

Kansas No. 4. Lump coal from mine of the Atchison Coal Mining Company, near Atchison, Kans.

Kansas No. 5. Lump and nut coal from mine No. 11 of the Southwestern Development Company, West Mineral, Kans.

Kentucky No. 1. Run-of-mine coal from Straight Creek mine No. 2 of the National Coal and Iron Company, Straight Creek, Ky.

Kentucky No. 2. First test: Lump, nut, pea, and slack coal from mine No. 11 of the St. Bernard Mining Company, Earlington, Ky.

Second test: Large briquettes containing 8 per cent of pitch. See p. 160.

Kentucky No. 3. Run-of-mine coal from Barnsley mine of the St. Bernard Mining Company, near Earlington, Ky.

Kentucky No. 4. Run-of-mine coal from mine or the Wheatcroft Coal and Mining Company, Wheatcroft, Ky.

Missouri No. 1. First test: Run-of-mine coal from New Home mine No. 1 of the New Home Coal Company, located at Sprague, Mo.

Second test: Large briquettes containing 11½ per cent of pitch. See p. 160.

Third test: Same as test No. 1.

Missouri No. 2. First test: Run-of-mine coal from mine No. 8 of the Northwestern Coal and Mining Company, Bevier, Mo.

Second test: Same as above.

Missouri No. 3. First test: Slack coal from mine of the Mendota Coal and Mining Company, Mendota, Mo.

Second test: Same as above except washed.

Missouri No. 4. Run-of-mine coal from mine of Morgan County Coal Company, near Barnett, Mo.

New Mexico No. 1. Lump and slack coal from Weaver mine of the American Fuel Company, 3 miles north of Gallup, N. Mex.

New Mexico No. 2. First test: Slack coal from Otero mine of the Caledonian Coal Company, 2 miles east of Gallup, N. Mex.

Second test: Small briquettes containing patent binder. See p. 162.

North Dakota No. 1. Run-of-mine brown lignite from Lehigh, N. Dak.

Pennsylvania No. 1. Coal from Eureka mine No. 31 of Berwind-White Coal Mining Company, Windber, Pa.

Pennsylvania No. 2. Coal from Eureka mine No. 31 of Berwind-White Coal Mining Company, Windber, Pa.

Pennsylvania No. 3. Small briquettes made with patent binder from anthracite culm furnished by Pennsylvania Coal Company, Scranton, Pa., p. 164.

West Virginia No. 1. Run-of-mine coal from mine of the Virginia and Pittsburg Coal Company, Kingmont, W. Va.

West Virginia No. 2. Run-of-mine coal from Pitcairn mine of the Pitcairn Coal Company, Clarksburg, W. Va.

West Virginia No. 3. Run-of-mine coal from mine of West Virginia Coal Company, Richard, W. Va.

West Virginia No. 4. Run-of-mine coal from mine of the West Virginia Coal Company, Bretz, W. Va.

West Virginia No. 5. Lump and nut coal from mine of the Davis Colliery Company, Coalton, W. Va.

West Virginia No. 6. First test. Run-of-mine coal from mine of the New River Smokeless Coal Company, Rushrun, W. Va.

Second test. Same as above.

West Virginia No. 7. Run-of-mine coal from mine of the New River Smokeless Coal Company, Sun, W. Va.

West Virginia No. 8. Run-of-mine coal from mine of the Gauley Mountain Coal Company, Ansted, W. Va.

West Virginia No. 9. Run-of-mine coal from Vulcan mine of the Mount Carbon Coal Company, Limited, Powellton, W. Va.

West Virginia No. 10. Lump and run-of-mine coal from Stuart M. Buck, Mora, W. Va.

West Virginia No. 11. Run-of-mine coal from mines Nos. 1 and 2 from W. H. Coffman, Zenith, Va.

West Virginia No. 12. First test. Run-of-mine coal from mine of the Big Sandy Coal and Coke Company, Big Sandy, W. Va.

Second test. Small briquettes made with patent binder.

Wyoming No. 1. Black lignite from mine of the Wyoming Coal and Mining Company, Monarch, Wyo.

Wyoming No. 2, Run-of-mine coal from Antelope Nos. 1 and 2 and Jumbo mines of the Cambria Fuel Company, Cambria, Wyo.

## RESULTS OF THE COAL TESTS UNDER THE BOILERS.

The accompanying table shows some of the most important practical results of the tests of coals burned under the boilers. In each test an effort was made to operate the boiler at a point very near its rated capacity. In some cases the coals tested presented difficulties which made it impossible to accomplish this result. The various coals were tested in the order of the "test numbers" given in column 1, and tests of coals from widely separated localities were frequently made on successive days.

Attention is called to "rate of combustion" at which the coals were burned. To those familiar with tests of this character it will be evident that the coals were for the most part burned at a rate calculated to give approximately the most favorable results.

All of the chemical results reported in this table were furnished by the chemical laboratory in charge of Prof. N. W. Lord, which is sufficient guarantee of the accuracy.

The "horsepower developed by the boiler," recorded in column 11, refers to the standard boiler horsepower and is the evaporation of 34.5

pounds of water per hour from a feed-water temperature of 212° F. and at atmospheric pressure.

In column 13 will be found recorded the number of pounds of water evaporated by one pound of dry coal at and from a temperature of 212° F. This column gives the best comparative results of the relative values of the coals tested, as far as these results relate to their commercial values.

The final report on these tests will give the values of all the items of the standard code report of the American Society of Mechanical Engineers and the complete logs of all tests, as well as a graphic chart of each trial.

*Results of coal tests under boilers.*

| Test No. | Name of sample. | Coal, size, and condition. | Chemical composition. | | | | | Duration of trial. | Total coal consumed. | Horsepower developed by boiler. | Dry coal burned per square foot of grate surface per hour. | Equivalent evaporation from and at 212° F. per pound of dry coal. | Dry coal per indicated horsepower hour. | Dry coal per electrical horsepower hour. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Fixed carbon. | Volatile matter. | Moisture. | Ash. | Sulphur (separately determined). | | | | | | | |
| 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 |
| | | | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Hours.* | *Lbs.* | | *Lbs.* | *Lbs.* | *Lbs.* | *Lbs.* |
| 17 | Alabama No. 1 | Nut, bright | 52.52 | 31.00 | 2.56 | 13.92 | 0.78 | 10.03 | 8,656 | 205.8 | 20.72 | 8.44 | 3.35 | 4.14 |
| 21 | .....do | Large briquettes | 50.96 | 33.00 | 2.63 | 13.41 | .94 | 8.25 | 6,521 | 196.6 | 18.97 | 8.81 | 3.21 | 3.96 |
| 16 | Alabama No. 2 | Mine run, bright | 48.65 | 32.98 | 4.83 | 13.54 | 1.17 | 10.02 | 9,198 | 216.4 | 21.54 | 8.55 | 3.31 | 4.08 |
| 9 | Arkansas No. 1 | Lump, bright | 66.36 | 18.61 | 1.99 | 13.04 | 1.21 | 10.07 | 7,071 | 180.5 | 16.90 | 9.05 | 3.12 | 3.86 |
| 14 | .....do | Large briquettes | 67.65 | 21.21 | .94 | 10.20 | 1.73 | 10.03 | 7,700 | 206.4 | 18.74 | 9.37 | 3.02 | 3.73 |
| 8 | Arkansas No. 2 | Lump, bright | 73.65 | 16.86 | 1.07 | 8.42 | 1.95 | 10.07 | 6,517 | 180.6 | 15.70 | 9.73 | 2.91 | 3.59 |
| 11 | .....do | Large briquettes | 60.30 | 22.49 | 4.88 | 12.33 | 1.32 | 9.98 | 7,370 | 190.3 | 17.31 | 9.31 | 3.00 | 3.71 |
| 29 | Arkansas No. 3 | Mine run, bright | 72.74 | 16.04 | 1.97 | 9.25 | 1.29 | 10.02 | 8,158 | 219.7 | 19.68 | 9.50 | 2.98 | 3.68 |
| 35 | .....do | Large briquettes | 62.04 | 17.35 | 2.60 | 18.01 | 1.41 | 10.08 | 8,600 | 200.8 | 20.49 | 8.34 | 3.39 | 4.19 |
| 40 | Arkansas No. 4 | .....do | 71.98 | 14.06 | 3.85 | 10.11 | 1.64 | 9.95 | 8,142 | 214.5 | 19.41 | 9.40 | 3.01 | 3.72 |
| 42 | .....do | Small briquettes | 66.03 | 18.60 | 3.18 | 12.19 | 1.70 | 7.03 | 5,238 | 166.8 | 17.78 | 7.98 | 3.54 | 4.38 |
| 41 | Arkansas No. 5 | Nut, bright | 73.68 | 12.54 | 2.22 | 11.56 | 2.39 | 10.13 | 9,000 | 202.4 | 21.42 | 8.04 | 3.52 | 4.34 |
| 75 | Colorado No. 1, lignite | .....do | 39.00 | 35.85 | 19.78 | 5.37 | .42 | 9.97 | 8,972 | 151.0 | 17.80 | 7.21 | 3.92 | 4.85 |
| 18 | Illinois No. 1 | Nut, dull | 38.21 | 36.91 | 9.69 | 15.19 | 4.40 | 9.93 | 11,124 | 211.4 | 24.90 | 7.21 | 3.92 | 4.85 |
| 19 | Illinois No. 2 | Slack, dull, washed | 41.72 | 37.77 | 10.45 | 10.06 | 3.36 | 9.97 | 10,096 | 210.2 | 22.36 | 8.00 | 3.54 | 4.38 |
| 38 | Illinois No. 3 | Nut, dull | 48.75 | 31.19 | 8.51 | 11.55 | 1.50 | 10.13 | 9,537 | 200.6 | 21.23 | 8.04 | 3.52 | 4.34 |
| 48 | Illinois No. 4 | Lump, dull | 41.59 | 33.48 | 13.47 | 11.46 | 1.28 | 10.07 | 9,367 | 172.0 | 19.84 | 7.37 | 3.83 | 4.73 |
| 50 | .....do | Nut, dull | 43.63 | 32.44 | 12.58 | 11.35 | 1.36 | 10.02 | 10,750 | 197.8 | 23.13 | 7.27 | 3.89 | 4.8 |
| 73 | Illinois No. 6 | .....do | 39.62 | 32.31 | 13.19 | 14.88 | 3.48 | 9.92 | 10,343 | 194.3 | 22.34 | 7.40 | 3.82 | 4.72 |
| 71 | Indiana No. 1 | Large briquettes | 43.23 | 38.79 | 11.74 | 6.24 | 2.03 | 6.60 | 6,579 | 222.6 | 21.70 | 8.73 | 3.24 | 4.00 |

| | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 68 | .....do | Nut, dull, washed | 40.41 | 35.17 | 16.59 | 7.83 | 2.18 | 9.93 | 10,819 | 222.7 | 22.39 | 8.45 | 3.84 | 4.13 |
| 65 | Indiana No. 2 | Nut, dull | 40.40 | 38.04 | 9.11 | 12.45 | 4.12 | 10.13 | 9,274 | 193.4 | 20.51 | 8.02 | 3.53 | 4.35 |
| 10 | Indian Territory No. 1 | Mine run, bright | 46.30 | 33.96 | 7.65 | 12.09 | 1.80 | 9.75 | 8,210 | 194.7 | 19.17 | 8.64 | 3.27 | 4.04 |
| 20 | Indian Territory No. 2 | Nut, bright | 50.31 | 36.21 | 3.71 | 9.77 | 1.39 | 10.17 | 9,213 | 212.9 | 21.50 | 8.42 | 3.36 | 4.15 |
| 32 | Indian Territory No. 3 | .....do | 47.58 | 37.30 | 4.79 | 10.33 | 3.93 | 10.02 | 9,145 | 205.9 | 21.43 | 8.17 | 3.46 | 4.27 |
| 52 | Indian Territory No. 4 | .....do | 45.83 | 35.44 | 6.24 | 12.99 | 3.86 | 10.22 | 9,296 | 186.1 | 21.04 | 7.58 | 3.76 | 4.64 |
| 45 | Iowa No. 1 | Nut, dull | 39.89 | 33.08 | 8.69 | 18.34 | 6.39 | 10.02 | 10,331 | 197.7 | 23.23 | 7.24 | 3.91 | 4.82 |
| 67 | Iowa No. 2 | Nut, very dirty | 33.73 | 35.35 | 14.88 | 16.04 | 4.73 | 9.92 | 10,986 | 192.7 | 23.28 | 7.05 | 4.01 | 4.95 |
| 49 | Iowa No. 3 | Nut, dull | 35.77 | 36.14 | 12.44 | 15.65 | 6.07 | 10.08 | 10,668 | 189.8 | 22.96 | 7.02 | 4.08 | 4.97 |
| 47 | Iowa No. 4 | Mine run, dull | 37.28 | 34.09 | 13.48 | 15.15 | 5.04 | 10.00 | 9,385 | 167.3 | 20.02 | 7.11 | 3.98 | 4.91 |
| 66 | .....do | Large briquettes | 37.85 | 36.50 | 13.24 | 12.41 | 3.90 | 10.03 | 9,900 | 184.5 | 21.11 | 7.43 | 3.80 | 4.70 |
| 55 | Iowa No. 5 | Nut, medium bright | 38.83 | 31.76 | 16.01 | 13.40 | 3.09 | 9.98 | 11,200 | 204.7 | 23.23 | 7.50 | 3.77 | 4.66 |
| 6 | Kansas No. 1 | Lump, dull | 49.46 | 33.78 | 5.90 | 10.86 | 3.82 | 10.10 | 7,222 | 158.2 | 16.60 | 8.11 | 3.49 | 4.31 |
| 7 | .....do | Mine run, dull | 48.57 | 32.68 | 4.80 | 13.95 | 4.94 | 9.70 | 7,492 | 184.0 | 18.13 | 8.63 | 3.28 | 4.05 |
| 5 | Kansas No. 2 | .....do | 46.68 | 31.23 | 4.18 | 17.91 | 6.27 | 7.00 | 5,600 | 181.5 | 18.90 | 8.17 | 3.46 | 4.28 |
| 13 | .....do | Nut, dull, washed | 51.22 | 34.32 | 5.82 | 8.64 | 3.77 | 9.38 | 8,111 | 209.8 | 20.08 | 8.89 | 3.18 | 3.93 |
| 3 | Kansas No. 3 | Mine run, dull | 50.99 | 33.52 | 2.03 | 13.46 | 5.39 | 9.91 | 7,700 | 169.6 | 18.77 | 7.69 | 3.68 | 4.55 |
| 4 | .....do | .....do | 51.05 | 34.30 | 2.25 | 12.40 | 4.80 | 10.40 | 7,101 | 148.8 | 16.45 | 7.69 | 3.68 | 4.55 |
| 58 | Kansas No. 4 | Nut bright | 43.59 | 36.32 | 5.51 | 14.58 | 8.46 | 10.05 | 7,835 | 165.5 | 18.18 | 7.75 | 3.65 | 4.51 |
| 72 | Kansas No. 5 | .....do | 51.36 | 32.42 | 4.31 | 11.91 | 4.40 | 9.90 | 8,651 | 215.2 | 20.62 | 8.88 | 3.18 | 3.93 |
| 60 | Kentucky No. 1 | Nut, very clean | 55.59 | 35.61 | 2.89 | 5.91 | 1.19 | 10.03 | 8,355 | 212.3 | 19.95 | 9.06 | 3.12 | 3.86 |
| 57 | Kentucky No. 2 | Nut, bright | 45.75 | 37.91 | 7.76 | 8.58 | 3.37 | 10.00 | 9,557 | 208.5 | 21.75 | 8.16 | 3.47 | 4.28 |
| 76 | .....do | Large briquettes | 44.32 | 37.07 | 7.11 | 11.50 | 3.71 | 9.82 | 9,371 | 207.1 | 21.87 | 8.06 | 3.51 | 4.33 |
| 64 | Kentucky No. 3 | Nut, bright | 44.84 | 37.32 | 7.92 | 9.92 | 3.91 | 10.07 | 9,645 | 211.5 | 21.75 | 8.27 | 3.42 | 4.22 |
| 62 | Kentucky No. 4 | .....do | 45.74 | 35.65 | 5.89 | 12.72 | 3.72 | 9.93 | 9,386 | 211.7 | 21.90 | 8.21 | 3.41 | 4.25 |
| 12 | Missouri No. 1 | Mine run, dull | 40.64 | 34.88 | 7.28 | 17.20 | 4.37 | 10.00 | 9,737 | 207.4 | 22.30 | 7.92 | 3.57 | 4.41 |
| 23 | .....do | Large briquettes | 41.85 | 37.60 | 6.38 | 14.17 | 4.56 | 5.10 | 4,500 | 191.2 | 20.37 | 7.99 | 3.54 | 4.37 |
| 15 | .....do | Nut, dull, washed | 44.21 | 36.81 | 7.93 | 11.05 | 3.58 | 9.91 | 9,041 | 207.0 | 20.71 | 8.50 | 3.33 | 4.11 |
| 37 | Missouri No. 2 | Nut, dull | 37.33 | 32.88 | 13.09 | 16.70 | 4.92 | 9.98 | 11,650 | 208.2 | 25.00 | 7.08 | 3.99 | 4.93 |
| 44 | .....do | .....do | 39.76 | 31.77 | 11.57 | 16.90 | 4.60 | 9.98 | 9,248 | 172.5 | 20.20 | 7.26 | 3.90 | 4.81 |

*Results of coal tests under boilers*—Continued.

| Test No. | Name of sample. | Coal, size, and condition. | Chemical composition. | | | | | Duration of trial. | Total coal consumed. | Horsepower developed by boiler. | Dry coal burned per square foot of grate surface per hour. | Equivalent evaporation from and at 212° F. per pound of dry coal. | Dry coal per indicated horsepower hour. | Dry coal per electrical horsepower hour. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Fixed carbon. | Volatile matter. | Moisture. | Ash. | Sulphur (separately determined). | | | | | | | |
| 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 |
| | | | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Hours.* | *Lbs.* | *Lbs.* | *Lbs.* | *Lbs.* | *Lbs.* | *Lbs.* |
| 78 | Missouri No. 3 | Nut, dull | 29.98 | 26.18 | 18.63 | 25.21 | 3.85 | 9.95 | 10,828 | 149.5 | 21.85 | 5.82 | 4.86 | 6.00 |
| 77 | .....do | Nut, dull, washed | 39.61 | 31.18 | 20.78 | 8.43 | 2.88 | 9.93 | 11,044 | 189.7 | 21.72 | 7.43 | 3.81 | 4.70 |
| 70 | Missouri No. 4 | Nut, dull | 42.11 | 40.10 | 12.24 | 5.55 | 4.98 | 9.98 | 9,515 | 214.1 | 20.64 | 8.83 | 3.20 | 3.96 |
| 27 | New Mexico No. 1 | Mine run, clean | 41.57 | 37.85 | 11.90 | 8.68 | .56 | 10.05 | 10,963 | 206.3 | 23.70 | 7.41 | 3.96 | 4.90 |
| 26 | New Mexico No. 2 | Pea, dull | 36.11 | 37.30 | 9.92 | 16.67 | 1.13 | 10.00 | 11,870 | 198.5 | 26.37 | 6.41 | 4.41 | 5.45 |
| 30 | .....do | Small briquettes | 39.07 | 37.56 | 6.75 | 16.62 | 1.61 | 8.12 | 7,765 | 185.8 | 22.00 | 7.19 | 3.93 | 4.86 |
| 33 | North Dakota No. 1, lignite | Nut, brown | 25.40 | 28.13 | 35.84 | 10.63 | 1.68 | 9.85 | 10,680 | 108.9 | 17.15 | 5.40 | 5.24 | 6.47 |
| 1 | Pennsylvania No. 1 | Mine run, bright | 75.69 | 15.80 | 1.10 | 7.41 | 1.49 | 9.88 | 6,365 | 166.9 | 15.70 | 9.04 | 3.11 | 3.84 |
| 2 | Pennsylvania No. 2 | .....do | 76.76 | 16.61 | .59 | 6.04 | .91 | 10.08 | 6,835 | 191.2 | 16.60 | 9.79 | 2.89 | 3.57 |
| 36 | Pennsylvania No. 3 | Small briquettes | 55.00 | 27.62 | 3.00 | 14.38 | 1.13 | 10.08 | 8,012 | 184.5 | 19.01 | 8.26 | 3.42 | 4.23 |
| 24 | West Virginia No. 1 | Nut, bright | 56.25 | 34.64 | 1.90 | 7.21 | .98 | 9.98 | 7,818 | 199.2 | 18.94 | 8.95 | 3.16 | 3.90 |
| 25 | West Virginia No. 2 | Mine run, bright | 48.80 | 39.23 | 2.01 | 9.96 | 2.71 | 10.22 | 8,354 | 212.4 | 19.75 | 9.14 | 3.09 | 3.82 |
| 28 | West Virginia No. 3 | Nut, bright | 56.11 | 30.31 | 2.54 | 11.04 | 1.38 | 9.97 | 8,225 | 211.6 | 19.82 | 9.08 | 3.11 | 3.84 |
| 31 | West Virginia No. 4 | .....do | 59.84 | 27.64 | 2.53 | 9.99 | .96 | 10.00 | 7,895 | 215.1 | 18.98 | 9.65 | 2.93 | 3.62 |
| 34 | West Virginia No. 5 | .....do | 58.66 | 28.95 | 2.11 | 10.28 | 1.01 | 10.08 | 7,700 | 207.8 | 18.44 | 9.59 | 2.95 | 3.81 |
| 39 | West Virginia No. 6 | .....do | 70.03 | 22.38 | 2.14 | 5.45 | .70 | 9.95 | 7,406 | 213.2 | 17.95 | 10.09 | 2.80 | 3.46 |
| 43 | .....do | .....do | 71.42 | 21.44 | 2.11 | 5.03 | .64 | 10.18 | 7,264 | 208.5 | 17.21 | 10.30 | 2.74 | 3.39 |
| 51 | West Virginia No. 7 | .....do | 68.27 | 20.23 | 2.68 | 8.82 | 1.52 | 10.18 | 7,700 | 210.1 | 18.15 | 9.85 | 2.87 | 3.55 |
| 58 | West Virginia No. 8 | .....do | 56.68 | 31.19 | 5.26 | 6.87 | .74 | 9.98 | 7,997 | 211.6 | 18.72 | 9.62 | 2.94 | 3.63 |
| 69 | West Virginia No. 9 | .....do | 59.47 | 31.11 | 3.42 | 6.00 | .82 | 10.00 | 7,464 | 210.8 | 17.78 | 10.09 | 2.80 | 3.46 |

| | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 54 | West Virginia No. 10 | .....do | 73.84 | 18.23 | 1.74 | 6.19 | .59 | 9.93 | 7,441 | 205.9 | 18.15 | 9.65 | 2.93 | 3.62 |
| 56 | West Virginia No. 11 | .....do | 68.36 | 16.31 | 4.85 | 10.48 | .47 | 9.97 | 7,700 | 213.7 | 18.13 | 10.03 | 2.82 | 3.48 |
| 46 | West Virginia No. 12 | .....do | 75.33 | 18.26 | 1.58 | 4.83 | .59 | 10.13 | 7,399 | 206.1 | 17.68 | 9.90 | 2.86 | 3.53 |
| 59 | .....do | Small briquettes | 67.46 | 24.02 | 2.32 | 6.20 | .84 | 10.25 | 7,515 | 211.8 | 17.66 | 10.20 | 2.77 | 3.42 |
| 63 | Wyoming No. 1 | Nut, bright | 31.61 | 40.56 | 21.81 | 6.02 | .63 | 10.15 | 11,946 | 186.6 | 22.69 | 7.00 | 4.04 | 4.99 |
| 61 | Wyoming No. 2 | Nut, very dirty | 34.58 | 35.55 | 11.10 | 18.77 | 3.87 | 9.95 | 12,028 | 184.3 | 26.51 | 5.92 | 4.77 | 5.90 |

## PERSONNEL.

The coal tests under steam boilers were all made under the direct supervision of Prof. D. T. Randall, of the Department of Mechanical Engineering at the University of Illinois. He was assisted by Messrs. J. J. Harman, H. B. Dirks, R. H. Kuss, H. Kreisinger, C. H. McClure, R. W. Rutt, C. H. Green, R. H. Post, and H. W. Weeks, the first three being replaced by the last three on September 1. Mr. Henry Arens was fireman for all the trials, and his work was intelligent and satisfactory. Mr. William Cameron fired the auxiliary boiler. The engine was under the charge of Mr. Otto Kinner, who was assisted by Mr. H. M. Horstmeier, oiler. The electrical apparatus was in charge of Mr. Jos. Underwood, who was assisted by Mr. A. P. Bridgeman, in charge of the switchboard.

The work of all the corps of observers was performed with an intelligent realization of the importance of the work and with painstaking accuracy.

# PRODUCER-GAS TESTS.

By Robert H. Fernald.

## INTRODUCTION.

In presenting this brief preliminary report of the producer-gas tests no attempt is made to give detailed information, as the complete report, to be published later, will deal at length with the methods employed in conducting the tests, details of apparatus used, and methods of working up the results from the data obtained.

## EQUIPMENT.

The plant installed is a Taylor pressure gas producer, furnished by R. D. Wood & Co., of Philadelphia. The producer of 250-horsepower capacity is known as a No. 7 gas producer. It is 8½ feet external diameter and 15 feet high and is connected through an economizer 3 feet in external diameter and 16 feet high to a scrubber, whose external dimensions are 8 feet in diameter by 20 feet in height. The scrubber is filled with gas-house coke, which is constantly flushed with water during the operation of the plant. From the scrubber the gas passes to the tar extractor, a piece of apparatus whose detailed construction is carefully guarded by the manufacturers of the producer, but which resembles in outward appearance a centrifugal pump. The speed of rotation of this device is of vital importance in tar extraction. After passing through the tar extractor the gas goes directly to the purifier, an iron box 8 feet square and 3 feet 3 inches in height. This box is filled with oxidized iron filings and shavings that remove the sulphur from the gas, which next passes to the holder, a receiver a little over 20 feet in diameter and 13 feet high, of 4,000 cubic feet capacity. From the holder the gas is conducted through a meter of 1,000,000,000 cubic feet capacity to a 3-cylinder vertical Westinghouse gas engine with cylinders of 19-inch diameter and 22-inch stroke, rated at 235 brake horsepower. The engine is in turn belted to a six-pole 175-kilowatt Westinghouse direct-current generator. The load on the generator is controlled by and the energy developed dissipated through water rheostats especially constructed for the purpose.

The instruments and apparatus used in connection with the tests were loaned by the manufacturers or dealers. These instruments are all standard, and were frequently calibrated at the National Bureau of Standards in the Electricity Building, World's Fair Grounds.

During the charging of the producer the coal was carefully sampled, the sample analyzed, and the calorific value of the coal determined at the chemical laboratory operated in connection with the plant. The coal analyses in the appended table were supplied by Prof. N. W. Lord, in charge of the chemical laboratory.

The calorimetric determinations from the gas and the gas analyses were made in an independent laboratory provided in the engine room near the producer plant.

Observations and readings were taken every twenty minutes during these tests, as were also the calorimetric determinations from the gas, but in the majority of tests the gas analyses were made once every two hours only.

## PERSONNEL.

During the first few weeks the gas producer was operated under the direct supervision of Mr. C. W. Lummis, a representative of R. D. Wood & Co. Mr. Lummis was followed by Mr. C. O. Nordenson, also from R. D. Wood & Co., who continued in charge of the operations of the producer to the close of the season. The gas engine has been run by Mr. J. G. Culbertson, a representative of the Westinghouse Machine Company.

The observations and computations have been made by a crew of college men, who have been trained in various technical institutions and who have supplemented this training with practical experience. The men employed in these two important departments are Messrs. R. W. Cummings, H. G. Ecker, H. A. Grine, M. H. Mount, R. E. Peshak, Kurt Toensfeldt, and W. C. Weidmann.

A double check system has been maintained throughout all computations, thus assuring a high degree of accuracy. The operating supervision of the tests has been under the direct control of Capt. John A. Laird, a consulting engineer of St. Louis.

## METHODS OF CONDUCTING TESTS.

The tests were begun on the basis of a total of fifty hours for each test. The plant was operated ten hours a day and then fires were banked for the night, the records being continued the next morning. This permitted one test a week only. With the small crew at command it seemed to be the best possible arrangement and was continued for the first two tests. It was then thought desirable to secure double the number of tests, and the schedule was arranged to conduct two tests per week, each of thirty consecutive hours, allowing sufficient time between tests to make the necessary change of fuel and to enable the fuel bed in the producer to be brought to a proper working condition.

As it was desired to test as many coals as possible in the few weeks remaining before the close of the Exposition, the highest possible economy was made a secondary consideration, and for a part of the time the plant was run with a leaky hopper and other unfortunate conditions, which naturally impaired its efficiency.

In comparing the results it should be borne in mind that in these preliminary tests the object has been to demonstrate the possibility of using these coals in a producer, and not to show how efficiently they could be burned. Although the results in many cases have been highly satisfactory, there is no question that in a second series of tests upon the same coals, made with the idea of showing the greatest economy, the amount of coal per horsepower per hour will, in the majority of cases, be much less.

During tests Nos. 5 to 14 inclusive the hopper of the gas producer leaked, and considerable gas was wasted, thus vitiating to a small but undetermined extent the efficiency results that might otherwise be shown for the coals tested during that period. But at the time of making these tests it was not practicable to stop the operations of the plant for repairs; and the main purpose of the preliminary tests being to determine whether the coals were suitable for producer-gas purposes, it was decided to proceed, in spite of the leak in the hopper, and to repeat later, under more favorable conditions, the tests for relative efficiency.

These points should be kept carefully in mind in examining the appended report, and criticism of the relatively poor showing of one coal as compared with another should be reserved until after the publication of the detailed report, which will give a full statement of the conditions under which each coal was tested and the influences that tended to make that particular test more or less favorable when compared with others.

The results show with what ease gas may be produced from bituminous coal and lignites, and, taken as a whole, indicate the satisfactory economic results that may be expected under ordinary working conditions.

Immediately after the close of the Exposition, it having been decided to continue the tests for some weeks longer, the plant was shut down in order to repair the leaking hopper and to prepare for cold weather. Operations were resumed on December 12 and continued until December 22, which is the close of the period covered by this report.

A glance at the logs of the tests shows very clearly the advisability of allowing a few hours for the manipulation of the plant after starting before beginning the official records of the tests. This is due to the fact that each test is made on a new coal, whose working qualities in the producer are entirely unknown. The operation of the gas engine upon gases from different coals requires careful study in order to secure the best mixture, proper point of ignition, etc. It has therefore been decided, should the operation of the plant be continued, that seventy-two consecutive hours shall be allowed for each test. During the first eight to twelve hours the operators of the producer and engine will devote their entire attention to securing the best possible manipulation of the plant, thus insuring sixty hours for obtaining results of high economy as well as testing the endurance of the plant.

## RESULTS OF THE TESTS.

The first official producer-gas test began October 3, 1904, and between that date and December 22 eighteen coals were tested. The results are given briefly in the following pages.

The brief notes appended to the reports of each test give an idea of the operating qualities of the coals from the standpoint of the expert in charge of the producer.

### COAL, ALABAMA NO. 2; PRODUCER-GAS TEST NO. 2.

[Coal from mine No. 5 of the Galloway Coal Company, Carbon Hill, Ala.; tested October 10, 11, 12, 13, 14, 1904.]

| | | |
|---|---|---|
| Duration of test | hours | 43 |
| Total coal consumed in producer | pounds | 13,350 |
| Moisture in coal | per cent | 3.76 |
| Dry coal consumed in producer | pounds | 12,848 |
| Refuse from dry coal | per cent | 9.85 |
| Total refuse from coal | pounds | 1,315 |
| Total combustible[a] consumed in producer | do | 12,035 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 310.5 |
| Dry coal consumed in producer | 299 |
| Combustible consumed in producer | 280 |
| Equivalent coal[b] used by producer plant | 341.4 |
| Equivalent dry coal used by producer plant | 328.7 |
| Equivalent combustible used by producer plant | 306.8 |

*British thermal units.*[c]

| | |
|---|---|
| Per pound of coal | 12,865 |
| Per pound of dry coal | 13,365 |
| Per pound of combustible | 14,820 |
| Per cubic foot of standard gas | 149.18 |
| From standard gas[d] per pound dry coal burned in producer | 9,000 |
| From standard gas per hour per brake horsepower | 11,420 |

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[a] "Combustible" is dry coal minus refuse.

[b] "Equivalent coal" is coal actually consumed in producer plus the coal equivalent of the steam used in operating the producer.

[c] A "British thermal unit" is the heat required to raise 1 pound of water from 62° F. to 63° F.

[d] *"Standard gas" is gas* at 62° F. and 14.7 pounds pressure.

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 775,500 |
| Per hour | 18,050 |
| Per pound coal consumed in producer | 58.1 |
| Per pound dry coal consumed in producer | 60.4 |
| Per pound combustible consumed in producer | 64.5 |
| Per pound equivalent coal used by producer plant | 52.9 |
| Per pound equivalent dry coal used by producer plant | 55 |
| Per pound equivalent combustible used by producer plant | 58.9 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 192.6 |
| Average electrical horsepower developed at switch board | 200.6 |
| Average brake horsepower † available for outside purposes | 226.6 |
| Average brake horsepower † developed at engine | 235.5 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.61 | 1.55 | 1.45 |
| Per electrical horsepower developed at switch board | 1.55 | 1.49 | 1.40 |
| Per brake horsepower † available for outside purposes | 1.37 | 1.32 | 1.23 |
| Per brake horsepower † developed at engine | 1.32 | 1.27 | 1.19 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 1.77 | 1.71 | 1.59 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.71 | 1.64 | 1.53 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 1.51 | 1.45 | 1.35 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 1.45 | 1.40 | 1.30 |

*Average composition of coal and gas.*

| COAL.* | Per cent. |
|---|---|
| Moisture | 3.76 |
| Volatile matter | 33.45 |
| Fixed carbon | 53.29 |
| Ash | 9.50 |
| Sulphur | .86 |

| GAS BY VOLUME. | Per cent. |
|---|---|
| Carbon dioxide | 8.16 |
| Oxygen | .10 |
| Carbon monoxide | 16.65 |
| Hydrogen | 7.20 |
| Methane | 5.64 |
| Nitrogen | 62.24 |
| | 99.99 |

The Alabama No. 2 is a clean, hard coal, which was fed in 1 to 1½ inch lumps, and worked well in the producer. It did not cake excessively and was easily handled. The greatest trouble in its use arose from its tendency to shrink away from the walls of the producer. It was necessary, therefore, to break it against the walls about twice every hour. The lumps did not swell and fuse together, but seemed to keep apart in the fire.

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† Based on an assumed efficiency of 85 per cent for generator and belt.

* All coal analyses given in this report were made under the direction of Prof. N. W. Lord, and they will appear in his final report.

### COAL, COLORADO NO. 1; PRODUCER-GAS TEST NO. 15.

[Coal from Simpson mine of the Northern Coal and Coke Company, Lafayette, Colo.; tested December 13, 14, 1904.]

| | | |
|---|---|---|
| Duration of test | hours | 30 |
| Total coal consumed in producer | pounds | 10,933 |
| Moisture in coal | per cent | 20.24 |
| Dry coal consumed in producer | pounds | 8,720 |
| Refuse from dry coal | per cent | 7.34 |
| Total refuse from coal | pounds | 640 |
| Total combustible consumed in producer | pounds | 8,080 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 364.4 |
| Dry coal consumed in producer | 290.7 |
| Combustible consumed in producer | 269.3 |
| Equivalent coal used by producer plant | 428.4 |
| Equivalent dry coal used by producer plant | 341.7 |
| Equivalent combustible used by producer plant | 316.6 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 9,767 |
| Per pound of dry coal | 12,245 |
| Per pound of combustible | 13,210 |
| Per cubic foot of standard gas | 149 |
| From standard gas, per pound dry coal burned in producer | 7,860 |
| From standard gas, per hour per brake horsepower | 9,700 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 460,300 |
| Per hour | 15,343 |
| Per pound coal consumed in producer | 42.1 |
| Per pound dry coal consumed in producer | 52.8 |
| Per pound combustible consumed in producer | 57 |
| Per pound equivalent coal used by producer plant | 35.8 |
| Per pound equivalent dry coal used by producer plant | 44.9 |
| Per pound equivalent combustible used by producer plant | 48.5 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 186.4 |
| Average electrical horsepower developed at switch board | 200.2 |
| Average brake horsepower † available for outside purposes | 219.3 |
| Average brake horsepower † developed at engine | 235.4 |

---

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.95 | 1.56 | 1.45 |
| Per electrical horsepower developed at switch board | 1.82 | 1.45 | 1.35 |
| Per brake horsepower† available for outside purposes | 1.66 | 1.32 | 1.23 |
| Per brake horsepower† developed at engine | 1.55 | 1.23 | 1.14 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.30 | 1.83 | 1.70 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.14 | 1.71 | 1.58 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes | 1.95 | 1.56 | 1.44 |
| Equivalent pounds used by producer plant per brake horsepower† developed at engine | 1.82 | 1.45 | 1.34 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 20.24 | Carbon dioxide | 10.11 |
| Volatile matter | 32.26 | Oxygen | .55 |
| Fixed carbon | 41.65 | Carbon monoxide | 17.38 |
| Ash | 5.85 | Hydrogen | 11.05 |
| | | Methane | 5.00 |
| Sulphur | .60 | Nitrogen | 55.90 |
| | | | 99.99 |

The Colorado No. 1 is a black lignite which clinkered badly in the producer, in spite of frequent poking, but the clinkers were not large and could be broken from the top of the producer. The gas was of a good, uniform quality, and there is no doubt that the fuel can be used to advantage in producers. It yielded a large amount of yellow "lignite tar."

### COAL, ILLINOIS NO. 3; PRODUCER-GAS TEST NO. 6.*

[Coal from mine No. 3 of the Southern Illinois Coal Mining and Washing Company, near Marion, Ill.; tested October 31 and November 1, 1904.]

| | | |
|---|---|---|
| Duration of test | hours | 30 |
| Total coal consumed in producer | pounds | 10,500 |
| Moisture in coal | per cent | 7.62 |
| Dry coal consumed in producer | pounds | 9,700 |
| Refuse from dry coal | per cent | 10.53 |
| Total refuse from coal | pounds | 1,021 |
| Total combustible consumed in producer | do | 8,679 |

---

* Gas-producer hopper leaked.

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 350 |
| Dry coal consumed in producer | 323.3 |
| Combustible consumed in producer | 289.3 |
| Equivalent coal used by producer plant | 386 |
| Equivalent dry coal used by producer plant | 356.7 |
| Equivalent combustible used by producer plant | 319.2 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 12,046 |
| Per pound of dry coal | 13,041 |
| Per pound of combustible | 14,560 |
| Per cubic foot of standard gas | 154.8 |
| From standard gas per pound dry coal burned in producer | 8,330 |
| From standard gas per hour per brake horsepower | 11,460 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 522,350 |
| Per hour | 17,412 |
| Per pound coal consumed in producer | 49.8 |
| Per pound dry coal consumed in producer | 53.9 |
| Per pound combustible consumed in producer | 60.2 |
| Per pound equivalent coal used by producer plant | 45.1 |
| Per pound equivalent dry coal used by producer plant | 48.8 |
| Per pound equivalent combustible used by producer plant | 54.5 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 192.5 |
| Average electrical horsepower developed at switch board | 199.6 |
| Average brake horsepower † available for outside purposes | 226.5 |
| Average brake horsepower † developed at engine | 235 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.82 | 1.68 | 1.50 |
| Per electrical horsepower developed at switch board | 1.75 | 1.62 | 1.45 |
| Per brake horsepower† available for outside purposes | 1.54 | 1.43 | 1.28 |
| Per brake horsepower† developed at engine | 1.49 | 1.38 | 1.23 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.01 | 1.85 | 1.66 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.93 | 1.79 | 1.60 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes | 1.70 | 1.58 | 1.41 |
| Equivalent pounds used by producer plant per brake horsepower† developed at engine | 1.64 | 1.52 | 1.36 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 7.62 | Carbon dioxide | 10.53 |
| Volatile matter | 30.87 | Oxygen | .15 |
| Fixed carbon | 51.78 | Carbon monoxide | 15.31 |
| Ash | 9.73 | Hydrogen | 8.35 |
| | | Methane | 4.46 |
| Sulphur | 1.69 | Nitrogen | 61.19 |
| | | | 99.99 |

The test on Illinois No. 3 coal was very satisfactory. The gas was good and the coal easily handled. There were no signs of clinkers in the fire, and it is certain that this coal could be used continuously without trouble from that source. It may be considered a good gas-producer coal.

### COAL, ILLINOIS NO. 4; PRODUCER-GAS TEST NO. 9.*

[Coal from mine No. 3 of the Donk Brothers Coal and Coke Company, Troy, Ill.; tested November 10 and 11, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 30 |
| Total coal consumed in producer | pounds.. | 10,500 |
| Moisture in coal | per cent.. | 12.43 |
| Dry coal consumed in producer | pounds.. | 9,190 |
| Refuse from dry coal | per cent.. | 10.53 |
| Total refuse from coal | pounds.. | 968 |
| Total combustible consumed in producer | do.... | 8,222 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 350 |
| Dry coal consumed in producer | 306.3 |
| Combustible consumed in producer | 274.1 |
| Equivalent coal used by producer plant | 398.2 |
| Equivalent dry coal used by producer plant | 348.5 |
| Equivalent combustible used by producer plant | 311.9 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 11,237 |
| Per pound of dry coal | 12,834 |
| Per pound of combustible | 14,344 |
| Per cubic foot of standard gas | 151.5 |
| From standard gas per pound dry coal burned in producer | 8,840 |
| From standard gas per hour per brake horsepower | 11,620 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 536,435 |
| Per hour | 17,881 |
| Per pound coal consumed in producer | 51.1 |
| Per pound dry coal consumed in producer | 58.4 |
| Per pound combustible consumed in producer | 65.3 |
| Per pound equivalent coal used by producer plant | 44.9 |
| Per pound equivalent dry coal used by producer plant | 51.4 |
| Per pound equivalent combustible used by producer plant | 57.4 |

---

*Gas-producer hopper leaked.

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 189.1 |
| Average electrical horsepower developed at switch board | 198.4 |
| Average brake horsepower* available for outside purposes | 223 |
| Average brake horsepower* developed at engine | 233 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.85 | 1.62 | 1.45 |
| Per electrical horsepower developed at switch board | 1.76 | 1.55 | 1.38 |
| Per brake horsepower* available for outside purposes | 1.57 | 1.37 | 1.23 |
| Per brake horsepower* developed at engine | 1.50 | 1.31 | 1.17 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.11 | 1.84 | 1.65 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.01 | 1.76 | 1.57 |
| Equivalent pounds used by producer plant per brake horsepower* available for outside purposes | 1.79 | 1.56 | 1.40 |
| Equivalent pounds used by producer plant per brake horsepower* developed at engine | 1.71 | 1.50 | 1.34 |

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 12.43 | Carbon dioxide | 9.72 |
| Volatile matter | 32.65 | Oxygen | .12 |
| Fixed carbon | 45.70 | Carbon monoxide | 15.12 |
| Ash | 9.22 | Hydrogen | 9.98 |
| | | Methane | 6.00 |
| Sulphur | 1.41 | Nitrogen | 59.06 |
| | | | 100.00 |

There was no trouble during the whole of the test of Illinois No. 4 coal in keeping the producer bed in good condition. The coal yielded a large amount of tar, and gave a rich, uniform gas, which left no trace of tar or sulphur in the engine valves. It may be placed among the better grade of gas-producer coals.

### COAL, INDIANA NO. 1; PRODUCER-GAS TEST NO. 14.†

[Coal from Mildred mine of the J. Woolley Coal Company, Mildred, Ind.; tested December 2 and 3, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 29.67 |
| Total coal consumed in producer | pounds.. | 11,700 |
| Moisture in coal | per cent.. | 11.51 |
| Dry coal consumed in producer | pounds.. | 10,360 |
| Refuse from dry coal | per cent.. | 11.33 |
| Total refuse from coal | pounds.. | 1,180 |
| Total combustible consumed in producer | pounds.. | 9,180 |

---

* Based on an assumed efficiency of 85 per cent for generator and belt.

† *Gas-producer hopper leaked.*

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 394.5 |
| Dry coal consumed in producer | 349.3 |
| Combustible consumed in producer | 309.5 |
| Equivalent coal used by producer plant | 434.6 |
| Equivalent dry coal used by producer plant | 384.8 |
| Equivalent combustible used by producer plant | 341.0 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 11,534 |
| Per pound of dry coal | 13,037 |
| Per pound of combustible | 14,720 |
| Per cubic foot of standard gas | 153.7 |
| From standard gas per pound dry coal burned in producer | 7,730 |
| From standard gas per hour per brake horsepower | 11,480 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 521,100 |
| Per hour | 17,560 |
| Per pound coal consumed in producer | 44.5 |
| Per pound dry coal consumed in producer | 50.3 |
| Per pound combustible consumed in producer | 56.7 |
| Per pound equivalent coal used by producer plant | 40.4 |
| Per pound equivalent dry coal used by producer plant | 45.6 |
| Per pound equivalent combustible used by producer plant | 51.5 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 188.3 |
| Average electrical horsepower developed at switch board | 199.9 |
| Average brake horsepower* available for outside purposes | 221.5 |
| Average brake horsepower* developed at engine | 235 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 2.10 | 1.86 | 1.64 |
| Per electrical horsepower developed at switch board | 1.97 | 1.75 | 1.55 |
| Per brake horsepower* available for outside purposes | 1.78 | 1.58 | 1.40 |
| Per brake horsepower* developed at engine | 1.68 | 1.49 | 1.32 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.31 | 2.04 | 1.81 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.17 | 1.93 | 1.71 |
| Equivalent pounds used by producer plant per brake horsepower* available for outside purposes | 1.96 | 1.74 | 1.54 |
| Equivalent pounds used by producer plant per brake horsepower* developed at engine | 1.85 | 1.64 | 1.45 |

* Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 11.51 | Carbon dioxide | 9.89 |
| Volatile matter | 36.04 | Oxygen | .25 |
| Fixed carbon | 42.37 | Carbon monoxide | 14.10 |
| Ash | 10.08 | Hydrogen | 9.56 |
| | | Methane | 6.08 |
| Sulphur | 2.61 | Nitrogen | 60.13 |
| | | | 100.01 |

The gas from Indiana No. 1 coal was of uniformly good quality. No trouble was experienced in working the bed, and the coal may be considered an excellent producer fuel. It yielded a fair amount of good black tar.

### COAL, INDIANA NO. 2; PRODUCER-GAS TEST NO. 13.*

[Coal from Electric mine of the T. D. Scales Coal Company, Boonville, Ind.; tested November 28 and 29, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 7 |
| Total coal consumed in producer | pounds.. | 2,100 |
| Moisture in coal | per cent.. | 8.72 |
| Dry coal consumed in producer | pounds.. | 1,917 |
| Refuse from dry coal | per cent.. | 10.66 |
| Total refuse from coal | pounds.. | 204.3 |
| Total combustible consumed in producer | do.... | 1,712.7 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 300 |
| Dry coal consumed in producer | 274 |
| Combustible consumed in producer | 244.8 |
| Equivalent coal used by producer plant | 338 |
| Equivalent dry coal used by producer plant | 312 |
| Equivalent conbustible used by producer plant | 278.8 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 11,822 |
| Per pound of dry coal | 12,953 |
| Per pound of combustible | 14,500 |
| Per cubic foot of standard gas | 159.3 |
| From standard gas per pound dry coal burned in producer | 10,140 |
| From standard gas per hour per brake horsepower | 11,750 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 122,160 |
| Per hour | 17,450 |
| Per pound coal consumed in producer | 58.2 |
| Per pound dry coal consumed in producer | 63.6 |
| Per pound combustible consumed in producer | 71.3 |
| Per pound equivalent coal used by producer plant | 51.6 |
| Per pound equivalent dry coal used by producer plant | 55.9 |
| Per pound equivalent combustible used by producer plant | 62.6 |

---

* Gas-producer hopper leaked.

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 191.4 |
| Average electrical horsepower developed at switch board | 201 |
| Average brake horsepower† available for outside purposes | 225 |
| Average brake horsepower† developed at engine | 236.5 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.57 | 1.43 | 1.28 |
| Per electrical horsepower developed at switch board | 1.49 | 1.36 | 1.22 |
| Per brake horsepower † available for outside purposes | 1.33 | 1.22 | 1.09 |
| Per brake horsepower † developed at engine | 1.27 | 1.16 | 1.03 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 1.77 | 1.63 | 1.46 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.68 | 1.55 | 1.39 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 1.52 | 1.39 | 1.24 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 1.43 | 1.32 | 1.18 |

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 8.72 | Carbon dioxide | 11.80 |
| Volatile matter | 39.60 | Oxygen | .07 |
| Fixed carbon | 41.95 | Carbon monoxide | 11.46 |
| Ash | 9.73 | Hydrogen | 10.60 |
| | | Methane | 6.10 |
| Sulphur | 4.23 | Nitrogen | 59.97 |
| | | | 100.00 |

The Indiana No. 2 coal did not yield particularly good results during the test. The gas produced from it was difficult to clean and contained a high percentage of sulphur. During the first part of the test the gas was low in heat value, but improved in quality after the fuel bed had been increased in depth. This shows that good results may be obtained from this coal if it is properly fired in the producer, and if adequate facilities are provided for scrubbing and purifying the gas.

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† Based on an assumed efficiency of 85 per cent for generator and belt.

## COAL, INDIAN TERRITORY NO. 1, PRODUCER-GAS TEST NO. 1.

[Coal from mine No. 1 of the Whitehead Coal and Mining Company, Henryetta, Ind. T.; tested October 3, 4, 5, and 6, 1904.]

| | |
|---|---|
| Duration of test ........ hours.. | 31 |
| Total coal produced in producer ........ pounds.. | 11,200 |
| Moisture in coal ........ per cent.. | 5 |
| Dry coal consumed in producer ........ pounds.. | 10,640 |
| Refuse from dry coal ........ per cent.. | 8.96 |
| Total refuse from coal ........ pounds.. | 955 |
| Total combustible consumed in producer ........ do.... | 9,685 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 361 |
| Dry coal consumed in producer | 344 |
| Combustible consumed in producer | 312 |
| Equivalent coal used by producer plant | 392.7 |
| Equivalent dry coal used by producer plant | 374 |
| Equivalent combustible used by producer plant | 339.3 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal | 12,787 |
| Per pound of dry coal | 13,455 |
| Per pound of combustible | 14,800 |
| Per cubic feet of standard gas | 159.15 |
| From standard gas per pound dry coal burned in producer | 8,620 |
| From standard gas per hour per brake horsepower | 12,350 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 577,000 |
| Per hour | 18,613 |
| Per pound coal consumed in producer | 51.6 |
| Per pound dry coal consumed in producer | 54.1 |
| Per pound combustible consumed in producer | 59.4 |
| Per pound equivalent coal used by producer plant | 47.4 |
| Per pound equivalent dry coal used by producer plant | 49.9 |
| Per pound equivalent combustible used by producer plant | 54.6 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 196 |
| Average electrical horsepower developed at switch board | 204 |
| Average brake horsepower † available for outside purposes | 230 |
| Average brake horsepower † developed at engine | 240 |

---

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes.. | 1.84 | 1.76 | 1.59 |
| Per electrical horsepower developed at switch board..... | 1.77 | 1.69 | 1.53 |
| Per brake horsepower† available for outside purposes... | 1.57 | 1.50 | 1.36 |
| Per brake horsepower† developed at engine............ | 1.50 | 1.43 | 1.30 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes............ | 2.00 | 1.91 | 1.73 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board................ | 1.92 | 1.83 | 1.66 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes ........... | 1.71 | 1.62 | 1.47 |
| Equivalent pounds used by producer plant per brake horsepower† developed at engine..................... | 1.64 | 1.56 | 1.41 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture........................ | 5.00 | Carbon dioxide................ | 8.25 |
| Volatile matter.................. | 36.51 | Oxygen......................... | .11 |
| Fixed carbon.................... | 49.98 | Carbon monoxide .............. | 19.39 |
| Ash ............................ | 8.51 | Hydrogen ...................... | 7.69 |
| | | Methane........................ | 4.92 |
| Sulphur ........................ | 1.43 | Nitrogen........................ | 59.65 |
| | | | 100.01 |

Indian Territory coal No. 1 is a moderately free-burning coal. It caked fairly well in the producer and made a good top crust, which was easily worked. It was charged into the producer after being crushed and screened over a half-inch screen. This was done while it was wet, and considerable slack was carried over with the lumps.

This coal may be considered a good producer coal.

### COAL, KENTUCKY NO. 3; PRODUCER-GAS TEST NO. 12.*

[Coal from Barnsley mine of the St. Bernard Mining Company, Earlington, Ky.; tested November 25 and 26, 1904.]

| | | |
|---|---|---|
| Duration of test........................................ | hours.. | 30 |
| Total coal consumed in producer......................... | pounds.. | 11,100 |
| Moisture in coal ....................................... | per cent.. | 7.28 |
| Dry coal consumed in producer........................... | pounds.. | 10,300 |
| Refuse from dry coal.................................... | per cent.. | 9.69 |
| Total refuse from coal.................................. | pounds.. | 1,000 |
| Total combustible consumed in producer ................. | do.... | 9,300 |

---

* Gas-producer hopper leaked.

### *Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 370 |
| Dry coal consumed in producer | 343.3 |
| Combustible consumed in producer | 310 |
| Equivalent coal used by producer plant | 410.8 |
| Equivalent dry coal used by producer plant | 381.2 |
| Equivalent combustible used by producer plant | 344.2 |

### *British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 12,283 |
| Per pound of dry coal | 13,226 |
| Per pound of combustible | 14,650 |
| Per cubic foot of standard gas | 155.9 |
| From standard gas per pound dry coal burned in producer | 8,610 |
| From standard gas per hour per brake horsepower | 12,540 |

### *Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 568,295 |
| Per hour | 18,943 |
| Per pound coal consumed in producer | 51.2 |
| Per pound dry coal consumed in producer | 55.1 |
| Per pound combustible consumed in producer | 61.1 |
| Per pound equivalent coal used by producer plant | 46.2 |
| Per pound equivalent dry coal used by producer plant | 49.7 |
| Per pound equivalent combustible used by producer plant | 55 |

### *Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 189.8 |
| Average electrical horsepower developed at switch board | 200.5 |
| Average brake horsepower† available for outside purposes | 223.5 |
| Average brake horsepower† developed at engine | 235.5 |

### *Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible.. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.95 | 1.81 | 1.63 |
| Per electrical horsepower developed at switch board | 1.85 | 1.71 | 1.55 |
| Per brake horsepower† available for outside purposes | 1.66 | 1.54 | 1.39 |
| Per brake horsepower† developed at engine | 1.57 | 1.46 | 1.32 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.16 | 2.01 | 1.81 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.05 | 1.91 | 1.72 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes | 1.84 | 1.71 | 1.54 |
| Equivalent pounds used by producer plant per brake horsepower† developed at engine | 1.75 | 1.62 | 1.46 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 7.28 | Carbon dioxide | 10.87 |
| Volatile matter | 38.57 | Oxygen | .29 |
| Fixed carbon | 45.16 | Carbon monoxide | 12.45 |
| Ash | 8.99 | Hydrogen | 10.92 |
| | | Methane | 6.52 |
| Sulphur | 3.86 | Nitrogen | 58.95 |
| | | | 100.00 |

The Kentucky No. 3 is a hard bituminous coal and burned in the producer with a hard top crust. It yielded good gas of uniform quality and clean black tar. It is well adapted for producer gas.

### COAL, MISSOURI NO. 2; PRODUCER-GAS TEST NO. 7.*

[Coal from mine No. 8 of the Northwestern Coal and Mining Company, Bevier, Mo.; tested November 3, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 4.33 |
| Total coal consumed in producer | pounds.. | 1,500 |
| Moisture in coal | per cent.. | 11.6 |
| Dry coal consumed in producer | pounds.. | 1,326 |
| Refuse from dry coal | per cent.. | 16.79 |
| Total refuse from coal | pounds.. | 223 |
| Total combustible consumed in producer | do.... | 1,103 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 346.5 |
| Dry coal consumed in producer | 306 |
| Combustible consumed in producer | 255 |
| Equivalent coal used by producer plant | 384.5 |
| Equivalent dry coal used by producer plant | 339.6 |
| Equivalent combustible used by producer plant | 283 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 10,505 |
| Per pound of dry coal | 11,882 |
| Per pound of combustible | 14,280 |
| Per cubic foot of standard gas | 140 |
| From standard gas per pound dry coal burned in producer | 8,820 |
| From standard gas per hour per brake horsepower | 11,560 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 83,545 |
| Per hour | 19,300 |
| Per pound coal consumed in producer | 55.7 |
| Per pound dry coal consumed in producer | 63 |
| Per pound combustible consumed in producer | 75.7 |
| Per pound equivalent coal used by producer plant | 50.2 |
| Per pound equivalent dry coal used by producer plant | 56.8 |
| Per pound equivalent combustible used by producer plant | 68.2 |

---

* Gas-producer hopper leaked.

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 185.7 |
| Average electrical horsepower developed at switch board | 198.6 |
| Average brake horsepower † available for outside purposes | 218.5 |
| Average brake horsepower † developed at engine | 233.5 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.87 | 1.65 | 1.37 |
| Per electrical horsepower developed at switch board | 1.74 | 1.54 | 1.28 |
| Per brake horsepower † available for outside purposes | 1.59 | 1.40 | 1.17 |
| Per brake horsepower † developed at engine | 1.48 | 1.31 | 1.09 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.07 | 1.83 | 1.52 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.94 | 1.71 | 1.43 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 1.76 | 1.55 | 1.30 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 1.65 | 1.45 | 1.21 |

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 11.60 | Carbon dioxide | 12.07 |
| Volatile matter | 35.28 | Oxygen | .20 |
| Fixed carbon | 38.28 | Carbon monoxide | 10.53 |
| Ash | 14.84 | Hydrogen | 7.63 |
| | | Methane | 6.33 |
| Sulphur | 4.56 | Nitrogen | 63.23 |
| | | | 99.99 |

In using Missouri No. 2 coal some difficulty was experienced in keeping the bed in good condition, owing to its tendency to break open in spots. The high percentage of sulphur in the coal did not add to its value as a producer fuel, but the writer's opinion is that it can be used to advantage in producers by increasing the facilities for purifying.

### COAL, MONTANA NO. 1; PRODUCER-GAS TEST NO. 5.*

[Coal from mine near Red Lodge, Mont.; tested October 24-25, 1904.]

| | | |
|---|---|---|
| Duration of test | hours | 22.33 |
| Total coal consumed in producer | pounds | 10,200 |
| Moisture in coal | per cent | 11.4 |
| Dry coal consumed in producer | pounds | 9,037 |
| Refuse from dry coal | per cent | 12.12 |
| Total refuse from coal | pounds | 1,095 |
| Total combustible consumed in producer | do | 7,942 |

---

* Gas-producer hopper leaked.

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 456.5 |
| Dry coal consumed in producer | 404.5 |
| Combustible consumed in producer | 355.8 |
| Equivalent coal used by producer plant | 506.8 |
| Equivalent dry coal used by producer plant | 449.1 |
| Equivalent combustible used by producer plant | 395 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 10,575 |
| Per pound of dry coal | 11,934 |
| Per pound of combustible | 13,580 |
| Per cubic foot of standard gas | 160.8 |
| From standard gas per pound of dry coal burned in producer | 6,580 |
| From standard gas per hour per brake horsepower | 11,340 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 369,500 |
| Per hour | 16,540 |
| Per pound coal consumed in producer | 36.23 |
| Per pound dry coal consumed in producer | 40.89 |
| Per pound combustible consumed in producer | 46.50 |
| Per pound equivalent coal used by producer plant | 32.64 |
| Per pound equivalent dry coal used by producer plant | 36.80 |
| Per pound equivalent combustible used by producer plant | 41.90 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 191 |
| Average electrical horsepower developed at switchboard | 199.5 |
| Average brake horsepower † available for outside purposes | 224.8 |
| Average brake horsepower † developed at engine | 234.7 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 2.39 | 2.12 | 1.86 |
| Per electrical horsepower developed at switch board | 2.29 | 2.03 | 1.78 |
| Per brake horsepower † available for outside purposes | 2.03 | 1.80 | 1.58 |
| Per brake horsepower † developed at engine | 1.95 | 1.72 | 1.52 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.65 | 2.35 | 2.07 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.54 | 2.25 | 1.98 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 2.26 | 2.00 | 1.76 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 2.16 | 1.91 | 1.68 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 11.40 | Carbon dioxide | 9.04 |
| Volatile matter | 34.55 | Oxygen | .36 |
| Fixed carbon | 43.31 | Carbon monoxide | 18.67 |
| Ash | 10.74 | Hydrogen | 8.00 |
| | | Methane | 4.84 |
| Sulphur | 1.72 | Nitrogen | 59.10 |
| | | | 100.01 |

The Montana No. 1 coal was a washed slack. It made good gas, but was too fine to be burned at the rated output of the producer. It clinkered badly, causing the blast to break through the bed in two or three places and make bad holes. The test is valuable principally as a demonstration that good gas can be made from coal of this class.

It is recommended that a water seal type of producer be used for fuel of this class, and also that the coal be fed in sizes not smaller than chestnut. In using lump coal the producer should be 20 per cent larger for the same horsepower than one running on ordinary coal; and for slack from 25 to 30 per cent larger.

After this test the bed of ashes was drawn from the producer, as the large clinkers would have been detrimental to the next following test.

### COAL, NORTH DAKOTA NO. 2; PRODUCER-GAS TEST NO. 10.*

[Coal from the mine of the Cedar Coulee Coal Company, near Williston, N. Dak.; tested November 14, 15, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 30 |
| Total coal consumed in producer | pounds.. | 13,800 |
| Moisture in coal | per cent.. | 39.56 |
| Dry coal consumed in producer | pounds.. | 8,340 |
| Refuse from dry coal | per cent.. | 10.53 |
| Total refuse from coal | pounds.. | 878 |
| Total combustible consumed in producer | do.... | 7,462 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 460 |
| Dry coal consumed in producer | 278 |
| Combustible consumed in producer | 249 |
| Equivalent coal used by producer plant | 510 |
| Equivalent dry coal used by producer plant | 308 |
| Equivalent combustible used by producer plant | 275.8 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 6,802 |
| Per pound of dry coal | 11,255 |
| Per pound of combustible | 12,600 |
| Per cubic foot of standard gas | 188.5 |
| From standard gas per pound dry coal burned in producer | 7,830 |
| From standard gas per hour per brake horsepower | 13,770 |

---

*Gas-producer hopper leaked.

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 346,400 |
| Per hour | 11,550 |
| Per pound coal consumed in producer | 25.15 |
| Per pound dry coal consumed in producer | 41.53 |
| Per pound combustible consumed in producer | 46.42 |
| Per pound equivalent coal used by producer plant | 22.68 |
| Per pound equivalent dry coal used by producer plant | 37.5 |
| Per pound equivalent combustible used by producer plant | 41.88 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 125.1 |
| Average electrical horsepower developed at switch board | 134.2 |
| Average brake horsepower† available for outside purposes | 147.3 |
| Average brake horsepower† developed at engine | 158 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 3.67 | 2.22 | 1.99 |
| Per electrical horsepower developed at switch board | 3.42 | 2.07 | 1.86 |
| Per brake horsepower† available for outside purposes | 3.13 | 1.89 | 1.69 |
| Per brake horsepower† developed at engine | 2.91 | 1.76 | 1.58 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 4.07 | 2.46 | 2.20 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 3.80 | 2.29 | 2.05 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes | 3.47 | 2.09 | 1.88 |
| Equivalent pounds used by producer plant per brake horsepower† developed at engine | 3.23 | 1.95 | 1.74 |

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 39.56 | Carbon dioxide | 8.69 |
| Volatile matter | 27.78 | Oxygen | .23 |
| Fixed carbon | 26.30 | Carbon monoxide | 20.90 |
| Ash | 6.36 | Hydrogen | 14.33 |
| | | Methane | 4.85 |
| Sulphur | .93 | Nitrogen | 51.02 |
| | | | 100.02 |

This brown lignite would be an ideal gas-producer fuel but for its tendency to clinker. It yielded a rich gas and not so very much tar. This tar was yellow and sticky and not at all like that from bituminous coal. The bed had to be carried deeper than for soft coal and had to be poked frequently to prevent the formation of clinkers. After thirty hours' continuous run the bed was in good condition, and the test could have been continued or a new run begun without renewing the producer bed.

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† Based on an assumed efficiency of 85 per cent for generator and belt.

During the test the engine carried only two-thirds of its normal load, but there seems to be no doubt that it could have carried full load throughout the entire run.

### COAL, TEXAS NO. 1; PRODUCER-GAS TEST NO. 11.*

[Coal from the mine of the Houston County Coal and Manufacturing Company, near Crockett, Tex.; tested November 21, 22, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 21.67 |
| Total coal consumed in producer | pounds.. | 12,800 |
| Moisture in coal | per cent.. | 33.50 |
| Dry coal consumed in producer | pounds.. | 8,510 |
| Refuse from dry coal | per cent.. | 15.85 |
| Total refuse from coal | pounds.. | 1,327 |
| Total combustible consumed in producer | do.... | 7,183 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 590 |
| Dry coal consumed in producer | 393 |
| Combustible consumed in producer | 332 |
| Equivalent coal used by producer plant | 660 |
| Equivalent dry coal used by producer plant | 439.5 |
| Equivalent combustible used by producer plant | 371.3 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 7,267 |
| Per pound of dry coal | 10,928 |
| Per pound of combustible | 12,945 |
| Per cubic foot of standard gas | 169.7 |
| From standard gas per pound dry coal burned in producer | 7,260 |
| From standard gas per hour per brake horsepower | 12,230 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 363,654 |
| Per hour | 16,800 |
| Per pound coal consumed in producer | 28.4 |
| Per pound dry coal consumed in producer | 42.7 |
| Per pound combustible consumed in producer | 50.6 |
| Per pound equivalent coal used by producer plant | 25.5 |
| Per pound equivalent dry coal used by producer plant | 38.2 |
| Per pound equivalent combustible used by producer plant | 45.3 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 187 |
| Average electrical horsepower developed at switch board | 198 |
| Average brake horsepower † available for outside purposes | 220 |
| Average brake horsepower † developed at engine | 233 |

---

* Gas-producer hopper leaked.

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes.. | 3.16 | 2.10 | 1.78 |
| Per electrical horsepower developed at switch board..... | 2.98 | 1.99 | 1.68 |
| Per brake horsepower † available for outside purposes ... | 2.68 | 1.79 | 1.51 |
| Per brake horsepower † developed at engine ............ | 2.54 | 1.69 | 1.43 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes............ | 3.53 | 2.35 | 1.99 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board............... | 3.34 | 2.22 | 1.88 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes ........... | 3.00 | 2.20 | 1.69 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine .................... | 2.83 | 1.99 | 1.60 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture........................ | 33.50 | Carbon dioxide.................. | 11.10 |
| Volatile matter.................. | 32.34 | Oxygen......................... | .22 |
| Fixed carbon ................... | 23.80 | Carbon monoxide ............... | 14.43 |
| Ash ............................ | 10.36 | Hydrogen....................... | 10.54 |
| | | Methane........................ | 7.48 |
| Sulphur ....................... | .63 | Nitrogen ....................... | 56.22 |
| | | | 99.99 |

The Texas No. 1 is a brown lignite strongly resembling that previously tested from North Dakota. The gas from it was not so rich as that from the North Dakota lignite, but it was higher in heat units than is the gas obtained from ordinary soft coal. The lignite was more difficult to handle in the producer than bituminous coal, but by frequent poking and by supplying the right amount of air to the producer the bed was kept in good condition, and at the end of the thirty-hour test it was possible to break up the clinkers in the bed, requiring the removal of only a few ashes before beginning a new test. This lignite yielded a large amount of tar of the same kind as the lignite previously tested. As a producer fuel it is better than many grades of bituminous coal.

### COAL, TEXAS NO. 2; PRODUCER-GAS TEST NO. 18.

[Coal from Consumers Lignite Company, Hoyt, Tex.; tested December 21, 22, 1904.]

| | | |
|---|---|---|
| Duration of test.............................................. | hours.. | 19.33 |
| Total coal consumed in producer ............................ | pounds.. | 9,050 |
| Moisture in coal ............................................. | per cent.. | 33.71 |
| Dry coal consumed in producer.............................. | pounds.. | 5,999 |
| Refuse from dry coal......................................... | per cent.. | 10.98 |
| Total refuse from coal........................................ | pounds.. | 658.7 |
| Total *combustible consumed in producer* ...................... | do.... | 5,340.3 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 468 |
| Dry coal consumed in producer | 310.3 |
| Combustible consumed in producer | 276.2 |
| Equivalent coal used by producer plant | 519.5 |
| Equivalent dry coal used by producer plant | 344.4 |
| Equivalent combustible used by producer plant | 306.6 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 7,348 |
| Per pound of dry coal | 11,086 |
| Per pound of combustible | 12,450 |
| Per cubic foot of standard gas | 156.2 |
| From standard gas per pound dry coal burned in producer | 8,060 |
| From standard gas per hour per brake horsepower | 10,570 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 309,140 |
| Per hour | 16,009 |
| Per pound coal consumed in producer | 34.2 |
| Per pound dry coal consumed in producer | 51.6 |
| Per pound combustible consumed in producer | 57.9 |
| Per pound equivalent coal used by producer plant | 30.8 |
| Per pound equivalent dry coal used by producer plant | 46.4 |
| Per pound equivalent combustible used by producer plant | 52.2 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 189.6 |
| Average electrical horsepower developed at switch board | 201.2 |
| Average brake horsepower † available for outside purposes | 223 |
| Average brake horsepower † developed at engine | 236.5 |

*Coal consumed, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 2.47 | 1.64 | 1.46 |
| Per electrical horsepower developed at switch board | 2.33 | 1.54 | 1.37 |
| Per brake horsepower † available for outside purposes | 2.10 | 1.39 | 1.24 |
| Per brake horsepower † developed at engine | 1.98 | 1.31 | 1.17 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.74 | 1.82 | 1.62 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.58 | 1.71 | 1.52 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 2.33 | 1.55 | 1.38 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 2.20 | 1.46 | 1.30 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 33.71 | Carbon dioxide | 9.60 |
| Volatile matter | 29.25 | Oxygen | .20 |
| Fixed carbon | 29.76 | Carbon monoxide | 18.22 |
| Ash | 7.28 | Hydrogen | 9.63 |
| | | Methane | 4.81 |
| Sulphur | .53 | Nitrogen | 57.53 |
| | | | 99.99 |

Texas No. 2 is a brown lignite that gave highly satisfactory results in the producer, yielding a rich, uniform gas and a large amount of yellow tar. It is an excellent fuel for producers.

### COAL, WEST VIRGINIA NO. 1; PRODUCER-GAS TEST NO. 3.

[Coal from the mine of the Virginia and Pittsburg Coal Company, Kingmont, W. Va.; tested October 17, 18, 1904.]

| | | |
|---|---|---|
| Duration of test | hours | 24 |
| Total coal consumed in producer | pounds | 6,900 |
| Moisture in coal | per cent | 1.61 |
| Dry coal consumed in producer | pounds | 6,790 |
| Refuse from dry coal | per cent | 6.24 |
| Total refuse from coal | pounds | 423.5 |
| Total combustible consumed in producer | do | 6,366.5 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 287.5 |
| Dry coal consumed in producer | 283 |
| Combustible consumed in producer | 265.5 |
| Equivalent coal used by producer plant | 320.6 |
| Equivalent dry coal used by producer plant | 315.6 |
| Equivalent combustible used by producer plant | 296.1 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal | 14,166 |
| Per pound of dry coal | 14,396 |
| Per pound of combustible | 15,350 |
| Per cubic foot of standard gas | 144.4 |
| From standard gas per pound dry coal burned in producer | 9,260 |
| From standard gas per hour per brake horsepower | 11,130 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 435,500 |
| Per hour | 18,150 |
| Per pound coal consumed in producer | 63.2 |
| Per pound dry coal consumed in producer | 64.1 |
| Per pound combustible consumed in producer | 68.4 |
| Per pound equivalent coal used by producer plant | 56.6 |
| Per pound equivalent dry coal used by producer plant | 57.5 |
| Per pound equivalent combustible used by producer plant | 61.3 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 190.1 |
| Average electrical horsepower developed at switchboard | 200.4 |
| Average brake horsepower† available for outside purposes | 223.8 |
| Average brake horsepower† developed at engine | 235.5 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.51 | 1.49 | 1.40 |
| Per electrical horsepower developed at switch board | 1.43 | 1.41 | 1.33 |
| Per brake horsepower† available for outside purposes | 1.29 | 1.27 | 1.19 |
| Per brake horsepower† developed at engine | 1.22 | 1.20 | 1.13 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 1.69 | 1.66 | 1.56 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.60 | 1.57 | 1.48 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes | 1.43 | 1.41 | 1.33 |
| Equivalent pounds used by producer plant per brake horsepower †developed at engine | 1.36 | 1.34 | 1.26 |

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 1.61 | Carbon dioxide | 10.50 |
| Volatile matter | 36.85 | Oxygen | .10 |
| Fixed carbon | 55.40 | Carbon monoxide | 14.34 |
| Ash | 6.14 | Hydrogen | 2.81 |
| | | Methane | 5.56 |
| Sulphur | .87 | Nitrogen | 66.69 |
| | | | 100.00 |

Previous to starting the test on this coal the tar extractor was removed and a new one substituted. As it was necessary to take the producer operator from his work to assist in placing the new extractor, the producer did not receive any attention for a few days and was in bad condition when the test was started. It was thought best to see if the producer could be made to carry full load and build up its bed into good running condition at the same time. The load was carried and the producer was in good shape for the official test six hours after being put into operation.

As the coal contained about 50 per cent slack, the producer was run with a hot bed, in order to coke the coal quickly after it fell on the fire. It made a good top coke, which was readily handled. The fire was easily managed and the coal may be considered an excellent fuel for producers.

---

†Based on an assumed efficiency of 85 per cent for generator and belt.

## COAL, WEST VIRGINIA NO. 4; PRODUCER-GAS TEST NO. 4.

[Coal from the mine of the West Virginia Coal Company, Bretz, W. Va.; tested October 20, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 9 |
| Total coal consumed in producer | pounds.. | 2,100 |
| Moisture in coal | per cent.. | 1.99 |
| Dry coal consumed in producer | pounds.. | 2,058.2 |
| Refuse from dry coal | per cent.. | 9 |
| Total refuse from coal | pounds.. | 184.9 |
| Total combustible consumed in producer | pounds.. | 1,873.3 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 233 |
| Dry coal consumed in producer | 229 |
| Combustible consumed in producer | 208 |
| Equivalent coal used by producer plant | 262.8 |
| Equivalent dry coal used by producer plant | 258.2 |
| Equivalent combustible used by producer plant | 234.5 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal | 13,918 |
| Per pound of dry coal | 14,202 |
| Per pound of combustible | 15,600 |
| Per cubic foot of standard gas | 143.2 |
| From standard gas per pound dry coal burned in producer | 11,610 |
| From standard gas per hour per brake horsepower | 11,320 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 167,000 |
| Per hour | 18,560 |
| Per pound coal consumed in producer | 79.6 |
| Per pound dry coal consumed in producer | 81.2 |
| Per pound combustible consumed in producer | 89.2 |
| Per pound equivalent coal used by producer plant | 70.6 |
| Per pound equivalent dry coal used by producer plant | 71.9 |
| Per pound equivalent combustible used by producer plant | 79.2 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 189.2 |
| Average electrical horsepower developed at switch board | 199.7 |
| Average brake horsepower† available for outside purposes | 222.5 |
| Average brake horsepower† developed at engine | 235 |

---

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.23 | 1.21 | 1.10 |
| Per electrical horsepower developed at switch board | 1.17 | 1.15 | 1.04 |
| Per brake horsepower † available for outside purposes | 1.05 | 1.03 | .935 |
| Per brake horsepower † developed at engine | .992 | .975 | .885 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 1.39 | 1.36 | 1.24 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.32 | 1.29 | 1.17 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 1.18 | 1.16 | 1.05 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 1.12 | 1.10 | .998 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 1.99 | Carbon dioxide | 10.16 |
| Volatile matter | 28.89 | Oxygen | .24 |
| Fixed carbon | 60.30 | Carbon monoxide | 15.82 |
| Ash | 8.82 | Hydrogen | 11.16 |
| | | Methane | 3.74 |
| Sulphur | .79 | Nitrogen | 58.88 |
| | | | 100.00 |

The West Virginia No. 4 coal is soft and friable, reaching the producer mainly as slack. It coked well on top of the bed and was readily worked. It promised to give the largest yield of gas of all the coals used thus far. Owing to the small supply of coal available the test was necessarily of short duration, but enough was done to show that it is an excellent gas-producer coal.

### COAL, WEST VIRGINIA NO. 9, PRODUCER-GAS TEST NO. 17.

[Coal from Vulcan mine of the Mount Carbon Coal Company (Limited), Powellton, W. Va.; tested December, 19, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 6.33 |
| Total coal consumed in producer | pounds.. | 1,900 |
| Moisture in coal | per cent.. | 2.66 |
| Dry coal consumed in producer | pounds.. | 1,848 |
| Refuse from dry coal | per cent.. | 5.89 |
| Total refuse from coal | pounds.. | 108.9 |
| Total combustible consumed in producer | pounds.. | 1,739.1 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 300 |
| Dry coal consumed in producer | 292 |
| Combustible consumed in producer | 274.9 |
| Equivalent coal used by producer plant | 328.9 |
| Equivalent dry coal used by producer plant | 320.1 |
| Equivalent combustible used by producer plant | 301.4 |

### *British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 14,195 |
| Per pound of dry coal | 14,580 |
| Per pound of combustible | 15,500 |
| Per cubic foot of standard gas | 151 |
| From standard gas per pound, dry coal burned in producer | 8,150 |
| From standard gas per hour per brake horsepower | 10,060 |

### *Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 99,781 |
| Per hour | 15,770 |
| Per pound coal consumed in producer | 52.6 |
| Per pound dry coal consumed in producer | 54 |
| Per pound combustible consumed in producer | 57.4 |
| Per pound equivalent coal used by producer plant | 48 |
| Per pound equivalent dry coal used by producer plant | 49.3 |
| Per pound equivalent combustible used by producer plant | 52.3 |

### *Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 186.9 |
| Average electrical horsepower developed at switch board | 201 |
| Average brake horsepower † available for outside purposes | 220 |
| Average brake horsepower † developed at engine | 236.5 |

### *Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.60 | 1.56 | 1.47 |
| Per electrical horsepower developed at switch board | 1.49 | 1.46 | 1.37 |
| Per brake horsepower † available for outside purposes | 1.36 | 1.33 | 1.25 |
| Per brake horsepower † developed at engine | 1.27 | 1.24 | 1.16 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 1.76 | 1.71 | 1.61 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.64 | 1.59 | 1.50 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 1.49 | 1.46 | 1.37 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 1.39 | 1.35 | 1.27 |

### *Average composition of coal and gas.*

| COAL. | Per cent. |
|---|---|
| Moisture | 2.66 |
| Volatile matter | 32.00 |
| Fixed carbon | 59.61 |
| Ash | 5.73 |
| Sulphur | 1.00 |

| GAS BY VOLUME. | Per cent. |
|---|---|
| Carbon dioxide | 10.40 |
| Oxygen | .20 |
| Carbon monoxide | 13.70 |
| Hydrogen | 9.55 |
| Methane | 6.60 |
| Nitrogen | 59.55 |
| | 100.00 |

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† Based on an assumed efficiency of 85 per cent for generator and belt.

The West Virginia No. 9 is a rich bituminous coal. It burned well and was easily handled in the producer, yielding a gas of uniformly good quality. It makes a good producer fuel. On account of an accident to the gas engine the test on this coal had to be stopped before it was completed. A second test is being made as this report goes to press. The results of the second test will be given in the final report.

### COAL, WEST VIRGINIA NO. 12, PRODUCER-GAS TEST NO. 8.*

[Coal from mine of the Big Sandy Coal and Coke Company, Big Sandy, W. Va.; tested November 7, 8, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 30 |
| Total coal consumed in producer | pounds.. | 8,100 |
| Moisture in coal | per cent.. | 1.43 |
| Dry coal consumed in producer | pounds.. | 7,984 |
| Refuse from dry coal | per cent.. | 6.54 |
| Total refuse from coal | pounds.. | 522 |
| Total combustible consumed in producer | do.... | 7,462 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 270 |
| Dry coal consumed in producer | 266.1 |
| Combustible consumed in producer | 248.7 |
| Equivalent coal used by producer plant | 304.9 |
| Equivalent dry coal used by producer plant | 300.5 |
| Equivalent combustible used by producer plant | 280.9 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 14,614 |
| Per pound of dry coal | 14,825 |
| Per pound of combustible | 15,860 |
| Per cubic foot of standard gas | 142.5 |
| From standard gas per pound dry coal burned in producer | 10,150 |
| From standard gas per hour per brake horsepower | 11,500 |

*Gas produced, cubic feet (reduced to standard).*

| | |
|---|---|
| Total | 568,700 |
| Per hour | 18,957 |
| Per pound coal consumed in producer | 70.2 |
| per pound dry coal consumed in producer | 71.2 |
| Per pound combustible consumed in producer | 76.2 |
| Per pound equivalent coal used by producer plant | 62.1 |
| Per pound equivalent dry coal used by producer plant | 63.2 |
| Per pound equivalent combustible used by producer plant | 67.5 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 191.3 |
| Average electrical horsepower developed at switch board | 199.8 |
| Average brake horsepower † available for outside purposes | 225 |
| Average brake horsepower † developed at engine | 235 |

---

* Gas-producer hopper leaked.

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 1.41 | 1.39 | 1.30 |
| Per electrical horsepower developed at switch board | 1.35 | 1.33 | 1.24 |
| Per brake horsepower† available for outside purposes | 1.20 | 1.18 | 1.11 |
| Per brake horsepower† developed at engine | 1.15 | 1.13 | 1.06 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 1.59 | 1.57 | 1.47 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 1.53 | 1.50 | 1.40 |
| Equivalent pounds used by producer plant per brake horsepower † available for outside purposes | 1.35 | 1.34 | 1.25 |
| Equivalent pounds used by producer plant per brake horsepower † developed at engine | 1.30 | 1.28 | 1.20 |

† Based on an assumed efficiency of 85 per cent for generator and belt.

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent. |
|---|---|---|---|
| Moisture | 1.43 | Carbon dioxide | 10.34 |
| Volatile matter | 18.93 | Oxygen | .12 |
| Fixed carbon | 73.19 | Carbon monoxide | 14.21 |
| Ash | 6.45 | Hydrogen | 12.98 |
| | | Methane | 4.61 |
| Sulphur | .95 | Nitrogen | 57.75 |
| | | | 100.01 |

The West Virginia No. 12 coal is of good quality and the bed was easily handled. It is soft and friable, and in going through the rolls much of it was reduced to a fine powder which gave trouble in the scrubber and wash box. It yielded considerable tar and may be considered a good gas-producer coal.

### COAL, WYOMING NO. 2, PRODUCER-GAS TEST NO. 16.

[Coal from mine of the Cambria Fuel Company, Cambria, Wyo.; tested December 16, 17, 1904.]

| | | |
|---|---|---|
| Duration of test | hours.. | 30 |
| Total coal consumed in producer | pounds.. | 12,100 |
| Moisture in coal | per cent.. | 9.44 |
| Dry coal consumed in producer | pounds.. | 10,958 |
| Refuse from dry coal | per cent.. | 22.9 |
| Total refuse from coal | pounds.. | 2,509 |
| Total combustible consumed in producer | do.... | 8,449 |

*Coal consumed, pounds per hour.*

| | |
|---|---|
| Coal consumed in producer | 403.3 |
| Dry coal consumed in producer | 365.3 |
| Combustible consumed in producer | 281.6 |
| Equivalent coal used by producer plant | 459.8 |
| Equivalent dry coal used by producer plant | 416.5 |
| Equivalent combustible used by producer plant | 321.1 |

*British thermal units.*

| | |
|---|---|
| Per pound of coal as fired | 9,650 |
| Per pound of dry coal | 10,656 |
| Per pound of combustible | 13.820 |
| Per cubic foot of standard gas | 151 |
| From standard gas per pound dry coal burned in producer | 6,168 |
| From standard gas per hour per brake horsepower | 9,516 |

*Gas produced (cubic feet reduced to standard).*

| | |
|---|---|
| Total | 447,700 |
| Per hour | 14,923 |
| Per pound coal consumed in producer | 37 |
| Per pound dry coal consumed in producer | 40.9 |
| Per pound combustible consumed in producer | 53 |
| Per pound equivalent coal used by producer plant | 32.5 |
| Per pound equivalent dry coal used by producer plant | 35.8 |
| Per pound equivalent combustible used by producer plant | 46.5 |

*Horsepower developed.*

| | |
|---|---|
| Average electrical horsepower available for outside purposes | 184.8 |
| Average electrical horsepower developed at switch board | 201.2 |
| Average brake horsepower† available for outside purposes | 217.4 |
| Average brake horsepower† developed at engine | 236.8 |

*Coal consumed in producer, pounds per horsepower per hour.*

| | Coal as fired. | Dry coal. | Combustible. |
|---|---|---|---|
| Per electrical horsepower available for outside purposes | 2.18 | 1.98 | 1.52 |
| Per electrical horsepower developed at switch board | 2.00 | 1.82 | 1.40 |
| Per brake horsepower† available for outside purposes | 1.86 | 1.68 | 1.30 |
| Per brake horsepower† developed at engine | 1.70 | 1.54 | 1.19 |
| Equivalent pounds used by producer plant per electrical horsepower available for outside purposes | 2.49 | 2.25 | 1.74 |
| Equivalent pounds used by producer plant per electrical horsepower developed at switch board | 2.28 | 2.07 | 1.60 |
| Equivalent pounds used by producer plant per brake horsepower† available for outside purposes | 2.11 | 1.92 | 1.48 |
| Equivalent pounds used by producer plant per brake horsepower† developed at engine | 1.94 | 1.76 | 1.36 |

*Average composition of coal and gas.*

| COAL. | Per cent. | GAS BY VOLUME. | Per cent |
|---|---|---|---|
| Moisture | 9.44 | Carbon dioxide | 10.21 |
| Volatile matter | 35.02 | Oxygen | .59 |
| Fixed carbon | 34.82 | Carbon monoxide | 15.46 |
| Ash | 20.72 | Hydrogen | 10.79 |
| | | Methane | 5.52 |
| Sulphur | 3.91 | Nitrogen | 57.43 |
| | | | 100.00 |

---

†Based on an assumed efficiency of 85 per cent for generator and belt.

|  |  |  | BRITISH THERMAL U |  |  |  |
|---|---|---|---|---|---|---|
|  |  | ucer | Per pound. |  |  | Per cubic foot of standard gas. |
| . of st. | Date | mbus-ble. | Coal as fired. | Dry coal. | Combustible. |  |
| 1 |  | 0 | 11 | 12 | 13 | 14 |
| 2 | Oct. | .8 | 12,865 | 13,365 | 14,820 | 149.2 |
| 15 | Dec. | .6 | 9,767 | 12,245 | 13,210 | 149.0 |
| " 6 | Oct. | .2 | 12,046 | 13,041 | 14,560 | 154.8 |
| " 9 | Nov. | .9 | 11,237 | 12,834 | 14,344 | 151.5 |
| " 14 | Dec. | .0 | 11,534 | 13,037 | 14,720 | 153.7 |
| " 13 | Nov. | .8 | 11,822 | 12,953 | 14,500 | 159.3 |
| 1 | Oct. | .3 | 12,787 | 13,455 | 14,800 | 159.2 |
| " 12 | Nov | 2 | 12,283 | 13,226 | 14,650 | 155.9 |
| " 7 | Nov | 0 | 10,505 | 11,882 | 14,280 | 140.0 |
| " 5 | Oct. | 0 | 10,575 | 11,934 | 13,580 | 160.8 |
| " 10 | Nov | 8 | 6,802 | 11,255 | 12,600 | 188.5 |
| " 11 | Nov | 3 | 7,267 | 10,928 | 12,945 | 169.7 |
| 18 | Dec. | 6 | 7,348 | 11,086 | 12,450 | 156.2 |
| 3 | Oct. | 1 | 14,166 | 14,396 | 15,350 | 144.4 |
| 4 | Oct. | 5 | 13,918 | 14,202 | 15,600 | 143.2 |
| 17 | Dec | 4 | 14,195 | 14,580 | 15,500 | 151.0 |
| " 8 | No | 9 | 14,614 | 14,825 | 15,860 | 142.5 |
| 16 | Dec | 1 | 9,650 | 10,656 | 13,820 | 151.0 |

The Wyoming No. 2 is a bituminous coal. It burned without any clinkering, leaving a large amount of white ash similar to that obtained from wood. The gas was not uniform in quality, on account of the difficulty experienced in keeping the bed in good condition. An unusually large amount of yellow tar was taken from the gas.

## SUMMARY OF RESULTS.

The table given herewith shows in a condensed form the results obtained in the producer-gas tests.

# COMPARISON OF RESULTS OBTAINED FROM STEAM AND PRODUCER-GAS TESTS.

By Robert H. Fernald and L. P. Breckenridge.

## STEAM PLANT.

The accompanying table, showing the comparative results of burning the various coals under the boiler and in the gas producer, is of much interest and value.

It is to be recollected that the steam generated by the boiler was used in a simple noncondensing engine of the Corliss type, whose "water rate" was 26.3 pounds of steam per hour per horsepower developed; that this engine was belted to the electric generator, and that the mechanical efficiency of this combination of engine and generator was 81 per cent.

With these figures available it will be an easy matter to calculate the number of pounds of coal which would have been required to produce an electrical horsepower provided a more economical type of steam engine had been used, or if the electrical generator had been directly connected to the engine, with the resulting advantage of a higher mechanical efficiency.

If, for example, the steam generated had been used by a steam engine capable of generating 1 horsepower with 18 pounds of steam per hour, and if the engine and generator had been direct connected, giving as high a mechanical efficiency as 90 per cent, then the "Total dry coal per electrical horsepower per hour" would have been reduced from 4.3 pounds, as given in column 13, to very nearly 3 pounds.

While these figures are frequently and easily attained by steam engines operating in large units, it will be conceded that in plants of from 200 to 250 horsepower they are but seldom reached.

It should be mentioned that the labor required would be the same for the operation of either the boiler plant or the gas-producer plant of the capacity under tests. In either plant two men would be sufficient.

*Comparative summary of the leading results of the coal tests made under the boiler and in the gas producer.*

| Name of sample. | Duration of trial. |  | Total dry coal consumed per hour.[a] |  | Dry coal burned per square foot of grate surface per hour. |  | Water evaporated from and at 212° F. per pound of dry coal. | B. T. U. per pound of dry coal used. |  | Electrical horsepower delivered to switch board. |  | Total dry coal per electrical horsepower per hour.[a] |  |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  | Steam plant. | Gas-producer plant. | Steam plant. | Gas-producer plant. | Steam plant. | Gas-producer plant.[b] | Steam plant. | Steam plant. | Gas-producer plant. | Steam plant. | Gas-producer plant. | Steam plant. | Gas-producer plant. |
|  | *Hours.* | *Hours.* | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* |  |  |  |  | *Pounds.* | *Pounds.* |
| Alabama No. 2 | 10.02 | 43.00 | 874 | 328.7 | 21.54 | 7.78 | 8.55 | 12,555 | 13,365 | 213.7 | 200.6 | 4.08 | 1.64 |
| Colorado No. 1 | 9.97 | 30.00 | 722 | 341.7 | 17.80 | 7.56 | 7.21 | 12,577 | 12,245 | 149.1 | 200.2 | 4.84 | 1.71 |
| Illinois No. 3 | 10.13 | 30.00 | 861 | 356.7 | 21.23 | 8.41 | 8.04 | 12,857 | 13,041 | 198.1 | 199.6 | 4.34 | [c]1.79 |
| Illinois No. 4 | 10.02 | 30.00 | 938 | 348.5 | 23.13 | 7.96 | 7.27 | 12,459 | 12,834 | 195.4 | 198.4 | 4.80 | [c]1.76 |
| Indiana No. 1 | 9.93 | 29.67 | 908 | 384.3 | 22.39 | 9.08 | 8.45 | 13,377 | 13,037 | 220.0 | 199.9 | 4.13 | [c]1.93 |
| Indiana No. 2 | 10.13 | 7.00 | 832 | 312.0 | 20.51 | 7.13 | 8.02 | 12,452 | 12,953 | 191.0 | 201.0 | 4.35 | [c]1.55 |
| Indian Territory No. 1 | 9.75 | 31.00 | 778 | 374.0 | 19.17 | 8.95 | 8.64 | 12,834 | 13,455 | 192.3 | 204.0 | 4.04 | 1.83 |
| Kentucky No. 3 | 10.07 | 30.00 | 882 | 381.2 | 21.75 | 8.92 | 8.27 | 13,036 | 13,226 | 208.9 | 200.5 | 4.22 | [c]1.91 |
| Missouri No. 2 | 9.98 | 4.33 | 1,014 | 339.6 | 25.00 | 7.96 | 7.08 | 11,500 | 11,882 | 205.6 | 198.6 | 4.93 | [c]1.71 |
| West Virginia No. 1 | 9.98 | 24.00 | 768 | 315.6 | 18.94 | 7.36 | 8.95 | 14,198 | 14,396 | 196.7 | 200.4 | 3.90 | 1.57 |
| West Virginia No. 4 | 10.00 | 9.00 | 770 | 258.2 | 18.98 | 5.96 | 9.65 | 14,002 | 14,202 | 212.5 | 199.7 | 3.62 | 1.29 |
| West Virginia No. 9 | 10.00 | 6.33 | 721 | 320.1 | 17.78 | 7.60 | 10.09 | 14,616 | 14,580 | 208.2 | 201.0 | 3.46 | 1.59 |
| West Virginia No. 12 | 10.13 | 30.00 | 719 | 300.5 | 17.68 | 6.92 | 9.90 | 15,170 | 14,825 | 203.6 | 199.8 | 3.53 | [c]1.50 |
| Wyoming No. 2 | 9.95 | 30.00 | 1,075 | 416.5 | 26.51 | 9.50 | 5.92 | 10,897 | 10,656 | 182.0 | 201.2 | 5.90 | 2.07 |

[a] In gas-producer plant this includes the coal consumed in the producer and the coal equivalent of the steam used in operating the producer.
[b] Coal actually consumed in producer only.
[c] Gas-producer hopper leaked during these tests.

## GAS-PRODUCER PLANT.

In considering the possible increase in efficiency of the boiler tests with a compound engine substituted for the simple engine used, the fact should not be overlooked that a corresponding increase in the efficiency of the gas-producer tests may be brought about under the most favorable conditions. The gas engine is passing through a transitional period. In the larger sizes the vertical single-acting engine is being replaced by the horizontal double-acting. Other changes and improvements are constantly being made which tend to do for the gas engine what compounding and tripling the expansions have already done for the steam engine.

The gas engine used in the trials recorded is a vertical three-cylinder, single-acting engine with no means of changing the ignition while the engine is running. A brief consideration of these points will lead at once to the conclusion that the gas engine and steam engine used in these tests compare very favorably, and that any increase in efficiency in the boiler tests that might result from using a compound engine can be offset by the introduction of the more modern type of gas engine.

# COKING TESTS.

By FRED. W. STAMMLER.

## INTRODUCTION.

The tests of the coking qualities of the coals received at the testing plant were made in a battery of three bee-hive ovens. The ovens are of standard size and shape, 12 feet in diameter and 7 feet high, and they were built from plans furnished by Mr. John Fulton, the well-known expert on coke making, of Johnstown, Pa. The writer was assisted in making the coking tests by Mr. B. B. Boyd, of Uniontown, Pa. The analyses given in the table at the end of this report, of coals that were charged in the coke ovens, and of the resulting cokes, were all made at the chemical laboratory of the testing plant under the direction of Prof. N. W. Lord, of Columbus, Ohio.

The ovens were fired on September 10, and were kept in blast until December 16. During this time 61 tests were made, embracing 44 different coals, coming from 11 States. Thirty-two tests were made on 29 different samples of raw coal, and 27 tests were made on 24 samples of washed coal. Two tests were made on briquettes of non-coking coal, having an additional amount of volatile matter used as a binding material. All of the coal charged into the ovens was first passed through rolls reducing it to 1½ inches or smaller.

## RESULTS OF TESTS.

The results of the coking tests have been arranged alphabetically by States, the general number adopted for all tests being given for ready identification, and are briefly summarized as follows:

*Alabama No. 1.*—Lump and nut coal from mine No. 8, Ivy Coal and Iron Company, Horse Creek, Ala.

Two charges were made of this coal, the first being unwashed and the second washed. The first charge of unwashed coal consisted of 8,000 pounds. Although the coking process was continued for 88 hours, the coke was very light, of a spongy structure, and high in ash. Washing reduced the percentage of impurities but slightly, though it is believed that with more thorough washing a coke of good quality might be obtained. The charge of washed coal weighed 9,800 pounds, was burned for 65 hours, and produced 5,731 pounds of coke,[a] a percentage yield of 58.5. This coke was also soft and high in ash.

*Alabama No. 2.*—Lump, nut, and pea coal from mine No. 5, Galloway Coal Company, Carbon Hill, Ala.

The coal used in this test was unwashed, the charge weighing 8,000 pounds. It coked imperfectly in small pieces, which were very soft, and which were mixed with charred coal and ash. The coal is high in ash (13.04 per cent), but if properly washed may yield a coke of fair grade.

*Arkansas No. 1.*—Lump and nut coal from mine No. 3, Central Coal and Coke Company, Huntington, Ark.

The charge consisted of 8,075 pounds of unwashed coal, which showed no tendency to coke and burned to ash. The volatile constituents are too low to make coke in the beehive oven. If properly washed this coal may make coke in a retort oven. It is high in ash (13.01 per cent) and also somewhat high in sulphur.

*Arkansas No. 2.*—Lump coal from mine No. 12, Central Coal and Coke Company, Bonanza, Ark. Same result as obtained from Arkansas No. 1.

*Arkansas No. 3.*—Lump and slack coal from mine No. 18, Western Coal and Mining Company, Jenny Lind, Ark. This was an experimental charge of a small amount of briquettes made from unwashed coal and placed in an oven with a coking coal. The briquettes contained 6 per cent of hard pitch and they produced a dense coke high in ash.

*Arkansas No. 6.*—Slack coal from mine No. 18, Western Coal and Mining Company, Jenny Lind, Ark.

The behavior of these two coals in the ovens was practically the same as that of Arkansas No. 1, except that No. 6 was washed and the percentage of ash shows a noticeable reduction.

The charge for a second test consisted of equal parts of unwashed coal crushed and mixed with 8 per cent of hard pitch, and the same material made into briquettes on the English machine. The addition of the pitch, by increasing the volatile matter, caused the coal to fuse and produced coke. The coke, however, was very dark and had large cells. The test is interesting, since it shows that a naturally noncoking coal may be made to coke by an addition of pitch. This charged weighed 8,000 pounds, was burned 42 hours, and yielded 4,056 pounds of coke and 539 pounds of breeze and ash.

*Illinois No. 1.*—Lump and nut coal from mine No. 1, Western Anthracite Coal and Coke Company, O'Fallon, Ill.

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[a] Exclusive of 352 pounds of breeze and ash.

This test was made on a charge of 9,000 pounds of unwashed coal, which was burned for 43 hours. It produced some small pieces of coke, mixed with charred coal and ash. The coal is too high in ash (15.95 per cent) and sulphur (4.14 per cent) to produce a coke fit for blast-furnace use.

*Illinois No. 2.*—Slack coal from the same mine as No. 1.

This coal was washed (p. 62) before charging into the ovens, reducing the ash to 9.19 per cent. The amount of coal charged was 9,000 pounds, which after burning 64 hours yielded 3,389 pounds of mixed hard and soft coke and 352 pounds of breeze and ash. The coke contained 20.18 per cent of ash and 2.75 per cent of sulphur.

*Illinois No. 3.*—Run-of-mine coal from mine No. 3, Southern Illinois Coal Mining and Washing Company, near Marion, Ill.

Two charges were made of this coal, one unwashed, the other washed. The unwashed charge of 9,000 pounds was tested for 43 hours. The coal lay dead in the oven, burning on top, but did not coke. The unwashed coal contained 10.59 per cent ash. The washed charge carried 5.86 per cent ash, being a reduction of nearly 50 per cent (p. 66). This charge of washed coal consisted of 13,000 pounds and was burned 90 hours, yielding 6,378 pounds of coke, which was very brittle and which broke up in handling into fine-fingered pieces. The breeze and ash in this oven weighed 834 pounds.

*Illinois No. 5.*—Grade No. 5, washed slack from mine No. 1, Donk Bros. Coal and Coke Company, Collinsville, Ill.

Three tests were made of this coal, as it was reported that it had been coked successfully in pits. All of the coal was washed before charging and each charge was burned for 42 hours. In the first test the coal was charged as it was received, but no coke was produced. In the second test the coal was rewashed through the New Century jig (p. 67), but it was charged wet and the oven was chilled and the coal would not burn. In the last test the oven was heated by an extra charge of coal and the charge was made in a red-hot oven, but even with this precaution the coal would not coke. In each instance the coal burned out on top, but otherwise lay dead in the oven and would not coke under existing conditions. All of the Illinois coals are too high in sulphur to make good blast-furnace coke.

*Indiana No. 1.*—Run-of-mine coal from Mildred mine, J. Woolley Coal Company, Mildred, Ind.

This coal was crushed and washed (p. 64). The charge consisted of 8,000 pounds and was burned for 40 hours. It produced 3,473 pounds of fair coke of medium weight, bright gray, but brittle and rather high in sulphur. The breeze and ash amounted to 368 pounds.

*Indian Territory No. 1.*—Lump and slack coal from mine No. 1, Whitehead Coal and Mining Company, Henryetta, Ind. T.

This test was made on 8,095 pounds of unwashed coal. It lasted 40 hours, and nothing but ash was left in the oven. The coal showed no tendency to coke.

*Indian Territory No. 2.*—Run-of-mine coal from mine No. 8, of Rock Island Coal Company, Hartshorne, Ind. T.

Two tests were made on this coal, washed and unwashed. Both charges weighed 9,000 pounds. The unwashed charge was burned for 66 hours, and yielded 5,725 pounds of coke and 580 pounds of breeze and ash. The coke was very soft, shattered and brittle, and somewhat high in sulphur. The washed charge contained one-third less ash than the unwashed (p. 67). It was burned for 65 hours, producing a coke which was hard and which had a fairly good ring. It showed considerable improvement in appearance over the coke made from unwashed coal, but the sulphur content was not materially reduced.

*Indian Territory No. 3.*—Run-of-mine coal from mine No. 1, D. Edwards & Sons, Edwards, Ind. T.

Two tests were made on this coal, the first being on 12,000 pounds of unwashed coal, burned 65 hours, and the second on 14,000 pounds washed coal, burned 66 hours. The washed coal, instead of showing a reduced percentage of sulphur, showed a gain (p. 68). Neither charge yielded coke. The ovens when drawn contained small pieces of charred coal and ash.

*Indian Territory No. 5.*—Slack and pea coal from mine No. 7, Western Coal and Mining Company, Lehigh, Ind. T.

This test was made on 10,000 pounds of washed coal and was continued for 65 hours (p. 68). It produced some small pieces of coke, fritted together, light and soft. The general conclusion regarding the Territory coals tested is that when properly washed they will produce coke, but not of a character suited for iron smelting, as the sulphur is too intimately combined or mixed with the coal to be removed by washing.

*Iowa No. 1.*—Lump and fine coal from mine No. 2, Anchor Coal Company, Laddsdale, Iowa.

In this test, as in all those on Iowa coals, the charge was of washed coal (p. 68). The charge weighed 9,500 pounds, and after burning 46 hours yielded 4,828 pounds of coke and 572 pounds of breeze and ash. The coke was brittle, with cracks lengthwise and crosswise through it. It was also high in sulphur and ash.

*Iowa No. 2.*—Run-of-mine coal from mine No. 5, Mammoth Vein Coal Company, Hamilton, Iowa.

The charge in this test consisted of 10,000 pounds of washed coal, which was burned for 64 hours (p. 69). The coke (3,866 pounds with 1,153 pounds of breeze and ash) was all in small pieces sintered together and with no bond.

*Iowa No. 3.*—Lump coal from mine No. 4, Gibson Coal Mining Company, Altoona, Iowa.

The charge in this test consisted of 8,000 pounds of washed (p. 69) coal, which was burned for 43 hours. It yielded 3,336 pounds of fine-fingered brittle coke that was high in sulphur and ash, and 585 pounds of breeze and ash.

*Iowa No. 4.*—Lump coal from mine No. 3, Centerville Block Coal Company, Centerville, Iowa.

The coke produced in this test was of the same general character as that obtained from Iowa No. 3, except that it was not quite so high in either sulphur or ash. The charge consisted of 8,000 pounds of washed (p. 69) coal, which was burned for 40 hours, producing 3,722 pounds of coke and 426 pounds of breeze and ash.

*Iowa No. 5.*—Run-of-mine coal from Inland mine No. 1, Inland Fuel Company, Chariton, Iowa.

The result of this test, made on 9,000 pounds of washed (p. 70) coal, and burned 66 hours, was a mixture of unburned coal, charred coke, and ash.

All of the Iowa coals tested are too high in sulphur to produce blast-furnace coke, and as the sulphur occurs largely as gypsum it can not be removed by washing. The ash is also high in relation to the fixed carbon.

*Kansas No. 1.*—Run-of-mine coal from mine No. 10, Western Coal and Mining Company, Fleming, Kans.

In making this test a charge of 11,300 pounds was made, and this was burned for 120 hours. At the end of the period the charge was found to be sintered on top, while under this sintering was a thin layer of light coke, with the remainder of the coal in the oven uncoked or burned.

*Kansas No. 2.*—Lump, nut, and slack coal from mine No. 11, Western Coal and Mining Company, Yale, Kans.

The behavior of this coal in the oven was similar to that of Kansas No. 1, showing no tendency to coke, although the charge of 9,125 pounds was burned for 159 hours.

*Kansas No. 3.*—Run-of-mine coal from mine No. 9, Southern Coal and Mercantile Company, Scammon, Kans.

This charge consisted of 7,100 pounds of unwashed coal, which was burned for 96 hours. It coked for about 12 inches from the top, while the remainder was simply charred coal and ash.

*Kansas No. 4.*—Lump coal from Atchison Coal Mining Company, Atchison, Kans.

This coal showed more of a tendency to coke than any of the other coals from Kansas. The charge, consisting of 10,000 pounds, was burned for 67 hours. It yielded 5,213 pounds of hard but brittle coke and 432 pounds of breeze and ash. The coke is, however, too high in sulphur (6.15 per cent) for use in an iron furnace, but it may be used in lead and zinc smelters, for which purpose a high percentage of sulphur is not detrimental.

*Kentucky No. 1.*—Run-of-mine coal from Straight Creek mine No. 2 of the National Coal and Iron Company, Straight Creek, Kentucky.

This test was made on 10,000 pounds of unwashed coal burned for 66 hours. It produced 5,441 pounds of hard coke, which was fine-fingered and easily broken into small pieces. The breeze and.ash weighed 355 pounds.

*Kentucky No. 3.*—Run-of-mine coal from Barnsley mine of St. Bernard Mining Company, Earlington, Ky.

In this test the charge consisted of 10,000 pounds of washed coal (p. 70), burned for 66 hours. It produced a coke of fair quality, of good color and ring, but somewhat high in sulphur. It produced 5,433 pounds of coke and 426 pounds of breeze and ash.

*Kentucky No. 4.*—Run-of-mine coal from Wheatcroft Coal and Mining Company, Wheatcroft, Ky.

This test was also made on 10,000 pounds washed coal (p. 70), which was burned for 66 hours, and yielded 5,558 pounds of coke and 313 pounds of breeze and ash. The coke was of good color and ring, but like Kentucky No. 4, was too high in sulphur for blast-furnace use.

*Missouri No. 2.*—Run-of-mine coal from mine No. 8 of the Northwestern Coal and Mining Company, Bevier, Mo.

This charge consisted of 12,000 pounds washed coal (p. 70), which was coked for 87 hours. It yielded 5,040 pounds of long-fingered brittle coke, having large pieces of slate mixed through it. The breeze and ash amounted to 580 pounds. The coke is high in both sulphur and ash.

*Missouri No. 3.*—Slack coal from Mendota Coal and Mining Company, Mendota, Mo.

This coal showed no tendency whatsoever to coke, although it was washed (p. 65) and burned for 42 hours.

*Missouri No. 4.*—Run-of-mine coal from Morgan County Coal Company, near Barnett, Mo.

This coal produced a coke of fair grade, of a gray color, and of a good metallic ring. The charge was 11,000 pounds of unwashed coal, and yielded 4,905 pounds of coke and 128 pounds of breeze and ash. The coke, however, is too high in sulphur for blast-furnace use, but it is believed that the quality of this coke could be improved by washing the coal before charging into the ovens.

*West Virginia No. 1.*—Run-of-mine coal, Virginia and Pittsburg Coal Company, Kingmount, W. Va.

This test was made on 9,000 pounds of unwashed coal. The charge was burned for 64 hours and yielded 5,572 pounds of coke and 268 pounds of breeze and ash. The coke was of good quality, but somewhat brittle.

*West Virginia No. 2.*—Run-of-mine coal from Pitcairn mine of the Pitcairn Coal Company, Clarksburg, W. Va.

Two tests were made of this coal, unwashed and washed. The first charge was 9,000 pounds of unwashed coal, producing 5,235 pounds of gray coke, somewhat brittle and high in sulphur and ash. The washed coal (p. 71), of which 13,000 pounds was charged, showed an improvement in the coke, though the sulphur was still too high for blast-furnace use. The yield from the second charge was 7,808 pounds, or 60 per cent, as compared with 58.2 per cent from the unwashed coal.

*West Virginia No. 3.*—Run-of-mine coal from West Virginia Coal Company, Richard, W. Va.

Two tests, one of unwashed and one of washed coal, were made on this sample. The charge of unwashed coal weighed 9,000 pounds and was coked 41 hours. It yielded 5,929 pounds of coke of gray color, brittle and somewhat high in ash, and 364 pounds of breeze and ash. The washed charge (p. 71) weighed 14,000 pounds, was coked for 66 hours, and yielded 9,070 pounds of coke of the same character as noted above and 535 pounds of ash and breeze. The quality of the coke was not improved by washing, except by a small reduction in the amount of ash.

The first test of 41 hours' duration gave a yield of 65.9 per cent, the second test of 66 hours gave a yield of 64.8 per cent.

*West Virginia No. 4.*—Run-of-mine coal from West Virginia Coal Company, Bretz, W. Va.

This coal was tested washed and unwashed. The unwashed charge contained 12,000 pounds. It was burned for 65 hours and yielded 8,907 pounds, or 74.2 per cent, of coke, and 428 pounds of breeze and ash. The charge of washed coal (p.71) weighed 9,000 pounds. It was coked for 42 hours and yielded 6,367 pounds, or 70.7 per cent, of coke, and 374 pounds of breeze and ash. The coke obtained was of good color, but was somewhat impaired by cross fractures. No improvement was noticed as a result of washing.

*West Virginia No. 5.*—Lump and nut coal from Davis Colliery Company, Coalton, W. Va.

Three coking tests were made on this shipment, one of unwashed and one of washed run-of-mine, crushed before coking to 1½ inches in diameter, and one of pulverized coal which was washed in the New Century jig (p. 72). The first charge of unwashed coal, weighing 13,000 pounds, was burned for 64 hours and yielded 8,298 pounds, or 63.8 per cent, of coke, and 497 pounds of ash and breeze. The washed charge, weighing 13,000 pounds, was burned 66 hours. It yielded 7,163 pounds, or 55.1 per cent, of coke, and 461 pounds of ash and breeze. The pulverized coal, weighing 10,000 pounds, was burned 69 hours and yielded 4,256 pounds, or 42.56 per cent, of coke, and 229 pounds of ash and breeze. The results showed that the coal would make a good coke if the ash could be materially reduced. Washing in this case lowered the percentage of ash from 19.14 per cent in the first test to 14.81 per cent in the second and 15.98 per cent in the third. The results of these tests show that the washing of this coal for coking purposes is advisable.

*West Virginia No. 6.*—Run-of-mine coal from the New River Smokeless Coal Company, Rush Run, W. Va.

Three tests were made on this coal, which was unwashed. The first charge, consisting of 13,000 pounds, was burned for 90 hours, and yielded 8,303 pounds, or 63.9 per cent, of light-gray, soft coke. The second charge, which consisted of 9,000 pounds, was burned for 66 hours, and yielded 6,399 pounds, or 71.1 per cent, of coke, of somewhat better quality than the first, although still gray and light. The charge consisted of 8,000 pounds of pulverized coal. It was burned for 44 hours, and yielded 5,849 pounds, or 73.1 per cent, of coke, which was dense and tough, but light in weight and color, and of poor ring.

*West Virginia No. 7.*—Run-of-mine coal from the New River Smokeless Coal Company, Sun, W. Va.

This test was made on unwashed coal, the charge weighing 8,000 pounds. It produced 5,119 pounds, or 64 per cent, of coke of good size and quality, but rather dark in color.

*West Virginia No. 8.*—Run-of-mine coal from Gauley Mountain Coal Company, Ansted, W. Va.

This test was made on 11,000 pounds of unwashed coal, which was burned for 66 hours and yielded 7,124 pounds, or 64.7 per cent, of coke, and 589 pounds of ash and breeze. The coke was hard, of good color, although of somewhat irregular cell structure.

*West Virginia No. 9.*—Run-of-mine coal from Vulcan mine of the Mount Carbon Coal Company (Limited), Powellton, W. Va.

Two tests were made on this coal, the first being unwashed, the second washed. The unwashed charge weighed 10,000 pounds, and was burned 40 hours. It yielded 6,084 pounds, or 60.8 per cent, of coke, and 314 pounds of breeze and ash. The coke was heavy and grayish in color. It was of good quality, though slightly brittle. The coke from the washed coal (p. 72) showed much improvement over that from the unwashed coal, and could be considered a high-grade coke. The charge of washed coal weighed 11,000 pounds, burned for 66 hours, and yielded 6,803 pounds, or 61.8 per cent, of coke, and only 78 pounds of ash and breeze.

*West Virginia No. 10.*—Lump and run-of-mine coal from Stuart M. Buck, Mora, W. Va.

This test was made on 11,000 pounds of unwashed coal. It was burned for 68 hours and yielded 7,858 pounds, or 71.4 per cent, of good, hard, heavy coke, slightly off color, and 429 pounds of ash and breeze.

*West Virginia No. 11.*—Run-of-mine coal from mines Nos. 1 and 2 of W. H. Coffman, Zenith, W. Va.

This was the only coal from West Virginia which did not coke in the ovens. The charge consisted of 11,000 pounds. It was burned for 67 hours, and when drawn consisted of ash mixed with raw, unburned coal.

*West Virginia No. 12.*—Run-of-mine coal from Big Sandy Coal and Coke Company, Big Sandy, W. Va.

Two tests of this coal were made, the first being unwashed and the second washed. The unwashed charge was 11,000 pounds. It was burned for 68 hours, and produced 7,082 pounds of coke, or 64.4 per cent, and 549 pounds of ash and breeze. The coke was of fair quality and had a metallic ring. It was of good color, but showed cross fractures. The washed charge (p. 73) of the same coal weighed 8,000 pounds, and was burned for 44 hours, producing 5,050 pounds, or 63.1 per cent, of coke and 583 pounds of ash and breeze. The coke was of large size and grayish color, but not particularly strong.

## CUPOLA TESTS OF COKE.

In connection with the coking tests of the coals the cokes made in the beehive ovens were not only analyzed and tested by various laboratory methods, but those which gave promise of value in foundry products were also tested in the cupolas of the model foundry connected with the Louisiana Purchase Exposition.

The testing of these cokes for foundry purposes was done under the supervision of a committee appointed for this purpose by the American Foundrymen's Association, the committee consisting of Dr. Richard Moldenke, secretary of the association; Mr. Herbert E. Field, of McIntosh, Hemphill & Co., Pittsburg, Pa., and Mr. W. J. Fogarty, of the Magnetite Foundry Company, St. Louis, Mo. Two thousand pounds of each such coke—one dozen samples in all—were reserved for these tests, and the cupola tests were made under the immediate supervision of Doctor Moldenke, during the month of November, using for this purpose two 36-inch small foundry cupolas, one of which was furnished by the Whiting Foundry Equipment Company, of Chicago, and the other by the J. S. McCormick Company, of Pittsburg.

The analyses of the cokes and the iron used in the tests have not yet been completed, however, and it is, therefore, thought best to delay the publication of the results of these tests until the issuance of a later report, in which will be included the tests of other cokes to be made during the next few months.

*Coking tests.*

| Name of sample. | Coal. | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Condition. | Chemical composition. | | | | | Amount charged in oven. | Coking time. |
| | | Moisture. | Volatile matter. | Fixed carbon. | Ash. | Sulphur. | | |
| | | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Pounds.* | *Hours.* |
| Alabama No. 1 | Raw | 2.65 | 30.79 | 52.68 | 13.88 | 0.76 | 8,000 | 88 |
| Do | Washed | 4.64 | 31.28 | 53.02 | 11.06 | 0.89 | 9,800 | 65 |
| Alabama No. 2 | Raw | 3.77 | 32.60 | 50.59 | 13.04 | 1.43 | 8,000 | 64 |
| Arkansas No. 1 | do | 2.31 | 18.10 | 66.58 | 13.01 | 1.13 | 8,075 | 114 |
| Arkansas No. 2 | do | 2.31 | 17.41 | 69.73 | 10.55 | 1.58 | 8,150 | 118 |
| Arkansas No. 6 | Washed | 18.29 | 12.95 | 62.54 | 6.22 | 1.22 | 9,000 | 95 |
| Arkansas No. 3 | Briquetted | ........ | ........ | ........ | ........ | ........ | Small box. | 65 |
| Arkansas No. 6 | do | 4.59 | 17.25 | 64.12 | 14.04 | 1.68 | 8,000 | 42 |
| Illinois No. 1 | Raw | 10.46 | 36.11 | 37.48 | 15.95 | 4.14 | 9,000 | 43 |
| Illinois No. 2 | Washed | 17.20 | 35.77 | 37.84 | 9.19 | 3.03 | 9,000 | 64 |
| Illinois No. 3 | Raw | 8.25 | 30.22 | 50.94 | 10.59 | 1.45 | 9,000 | 43 |
| Do | Washed | 9.52 | 32.05 | 52.57 | 5.86 | 1.41 | 13,000 | 90 |
| Illinois No. 5 | do | 17.86 | 31.65 | 32.93 | 17.56 | 3.25 | 8,000 | 42 |
| Illinois No. 5 | do | 25.15 | 28.68 | 36.99 | 9.18 | 2.71 | 5,000 | 42 |
| Illinois 5 | Washed | 14.60 | 30.76 | 36.48 | 18.16 | 3.44 | 9,000 | 42 |
| Indiana No. 1 | do | 16.53 | 36.38 | 40.05 | 7.04 | 2.03 | 8,000 | 40 |
| Indian Territory No. 1 | Raw | 10.29 | 31.46 | 44.39 | 13.86 | 1.34 | 8,095 | 40 |
| Indian Territory No. 2 | do | 3.82 | 37.45 | 48.74 | 9.99 | 1.47 | 9,000 | 66 |
| Do | Washed | 4.45 | 38.18 | 51.04 | 6.33 | 1.43 | 9,000 | 65 |
| Indian Territory No. 3 | Raw | 4.16 | 36.66 | 49.43 | 9.75 | 3.16 | 12,000 | 65 |
| Do | Washed | 5.93 | 38.05 | 48.53 | 7.49 | 3.20 | 14,000 | 66 |
| Indian Territory No. 5 | do | 16.95 | 33.30 | 41.61 | 8.14 | 2.90 | 10,000 | 65 |
| Iowa No. 1 | do | 12.84 | 35.91 | 41.00 | 10.25 | 4.61 | 9,500 | 46 |
| Iowa No. 2 | do | 12.85 | 35.44 | 35.43 | 10.28 | 3.93 | 10,000 | 64 |
| Iowa No. 3 | do | 16.83 | 39.27 | 35.87 | 8.03 | 4.55 | 8,000 | 43 |
| Iowa No. 4 | do | 17.80 | 37.59 | 37.39 | 7.14 | 3.59 | 8,000 | 40 |
| Iowa No. 5 | do | 19.25 | 31.07 | 41.75 | 7.93 | 2.28 | 9,000 | 66 |
| Kansas No. 1 | Raw | 8.10 | 31.26 | 45.82 | 14.84 | 4.48 | 11,300 | 120 |
| Kansas No. 2 | do | 3.96 | 30.46 | 45.54 | 20.04 | 5.59 | 9,125 | 159 |
| Kansas No. 3 | do | 1.75 | 32.95 | 48.39 | 16.91 | 5.60 | 7,100 | 96 |
| Kansas No. 4 | do | 6.63 | 36.99 | 43.45 | 12.93 | 7.19 | 10,000 | 67 |
| Kentucky No. 1 | do | 2.71 | 37.22 | 56.17 | 3.90 | 1.23 | 10,000 | 66 |
| Kentucky No. 3 | Washed | 10.51 | 37.33 | 44.76 | 7.40 | 2.51 | 10,000 | 66 |
| Kentucky No. 4 | do | 6.18 | 39.07 | 48.70 | 6.05 | 2.74 | 10,000 | 66 |
| Missouri No. 2 | do | 14.14 | 35.53 | 42.57 | 7.76 | 3.24 | 12,000 | 87 |
| Missouri No. 3 | do | 24.15 | 33.10 | 35.51 | 7.24 | 2.74 | 9,000 | 42 |
| Missouri No. 4 | Raw | 12.04 | 41.35 | 41.34 | 5.27 | 5.14 | 11,000 | 65 |
| West Virginia No. 1 | Raw | 1.93 | 36.50 | 54.97 | 6.60 | .88 | 9,000 | 64 |
| West Virginia No. 2 | do | 1.73 | 39.20 | 50.85 | 8.22 | 3.38 | 9,000 | 50 |
| Do | Washed | 3.98 | 39.76 | 49.21 | 7.06 | 2.84 | 13,000 | 64 |
| West Virginia No. 3 | Raw | 2.25 | 30.60 | 57.40 | 9.75 | .99 | 9,000 | 41 |
| Do | Washed | 4.11 | 29.54 | 57.34 | 9.01 | 1.18 | 14,000 | 66 |
| West Virginia No. 4 | Raw | 2.72 | 27.62 | 59.16 | 10.50 | .86 | 12,000 | 65 |
| Do | Washed | 3.47 | 27.95 | 61.05 | 7.53 | .74 | 9,000 | 42 |
| West Virginia No. 5 | Raw | 2.43 | 28.30 | 58.55 | 10.73 | .90 | 13,000 | 64 |
| Do | Washed | 4.84 | 28.12 | 56.76 | 10.28 | .91 | 13,000 | 66 |
| Do | Washed, fine. | 19.50 | 23.76 | 48.55 | 8.19 | .79 | 10,000 | 69 |

*Coking tests.*

| Coke. | | | | | | | | | | Remarks. |
|---|---|---|---|---|---|---|---|---|---|---|
| Amount produced. | Breeze and ash. | Chemical composition. | | | | | | Specific gravity. | Per cent produced. | |
| | | Moisture. | Volatile matter. | Fixed carbon. | Ash. | Sulphur. | Phosphorus. | | | |
| *Lbs.* | *Lbs.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | | | |
| | | | | | | | | | | Light, spongy. |
| 5,731 | 352 | 0.33 | 0.72 | 82.63 | 16.32 | 0.69 | 0.047 | 1.94 | 58.5 | Improved by washing. |
| | | | | | | | | | | Charred coal and ash. |
| | | | | | | | | | | No coke produced. |
| | | | | | | | | | | Do. |
| | | | | | | | | | | Do. |
| | | 1.05 | 2.80 | 72.73 | 23.42 | 1.52 | | | | Placed in oven with charge of coal. |
| 4,056 | 539 | 4.06 | 2.77 | 76.63 | 16.54 | 1.43 | .050 | 1.88 | 50.7 | 50 per cent briquettes and 50 per cent coal charged. |
| | | | | | | | | | | Small coke and ash. |
| 3,389 | 352 | 1.57 | 2.83 | 75.42 | 20.18 | 2.75 | .030 | 1.87 | 37.6 | Hard and soft mixed. |
| | | | | | | | | | | No coke, burned on top. |
| 6,378 | 834 | 6.11 | .42 | 82.55 | 10.92 | 1.13 | .019 | 1.84 | 49.1 | Fingered, brittle. |
| | | | | | | | | | | No coke produced. |
| | | | | | | | | | | Do. |
| | | | | | | | | | | Do. |
| 3,473 | 368 | 5.71 | 1.18 | 80.52 | 12.59 | 1.69 | .016 | 1.86 | 43.4 | Fair coke, brittle. |
| | | | | | | | | | | Burned to ash. |
| 5,725 | 580 | 8.15 | 1.74 | 75.68 | 11.43 | 1.50 | .050 | 1.88 | 63.6 | Coke soft. |
| 4,680 | 255 | .96 | 2.59 | 85.33 | 11.12 | 1.75 | .043 | 1.84 | 52.0 | Coke hard. |
| | | | | | | | | | | Small coke and ash. |
| | | | | | | | | | | Not improved. |
| | | | | | | | | | | Small coke fritted. |
| 4,828 | 572 | 10.53 | 1.65 | 70.39 | 17.45 | 3.89 | .051 | 1.87 | 50.7 | Very brittle. |
| 3,866 | 1,153 | | | | | | | | 38.7 | Sintered. |
| 3,336 | 585 | 5.73 | 1.87 | 75.49 | 16.91 | 4.57 | .0183 | 1.88 | 41.7 | Fine fingered. |
| 3,722 | 426 | 13.05 | 2.32 | 73.10 | 11.53 | 2.97 | .0123 | 1.82 | 46.5 | Do. |
| | | | | | | | | | | Charred coke and ash. |
| | | | | | | | | | | No coke produced. |
| | | | | | | | | | | Do. |
| | | | | | | | | | | Coked 12 inches from top. |
| 5,213 | 432 | .52 | 1.68 | 79.82 | 17.98 | 6.15 | .0148 | 1.92 | 52.1 | Hard and brittle. |
| 5,441 | 355 | 1.48 | .83 | 92.34 | 5.35 | .84 | .034 | 1.83 | 54.4 | Hard, fine fingered. |
| 5,433 | 426 | .14 | .56 | 86.31 | 12.99 | 2.16 | .009 | 1.81 | 54.3 | Fair coke. |
| 5,528 | 313 | .52 | .63 | 86.50 | 12.35 | 2.37 | .015 | 1.84 | 55.3 | Good color. |
| 5,040 | 580 | 3.45 | 1.80 | 80.27 | 14.48 | 2.79 | .02 | 1.86 | 42.0 | Long fingered. |
| | | | | | | | | | | No coke produced. |
| 4,905 | 128 | 2.51 | 1.11 | 85.57 | 10.81 | 4.60 | .018 | 1.83 | 44.6 | Fair coke. |
| 5,572 | 268 | .40 | 1.95 | 87.47 | 10.18 | .71 | .029 | 1.94 | 61.9 | Good coke. |
| 5,235 | 286 | .42 | .68 | 83.95 | 14.95 | 3.40 | .023 | 1.94 | 58.2 | Gray, brittle. |
| 7,808 | 302 | .59 | 1.31 | 86.70 | 11.40 | 2.24 | .017 | 1.89 | 60.1 | Improved by washing. |
| 5,929 | 364 | .22 | .83 | 80.77 | 18.18 | .93 | .065 | 1.95 | 65.9 | Gray coke. |
| 9,070 | 535 | .38 | .87 | 84.48 | 14.27 | 1.19 | .079 | 1.94 | 64.8 | Not improved by washing. |
| 8,907 | 428 | .62 | 1.43 | 86.10 | 11.85 | .82 | .031 | 1.84 | 74.2 | Good coke. |
| 6,367 | 374 | .20 | 1.15 | 85.42 | 13.23 | .69 | .019 | 1.94 | 70.7 | Not improved by washing. |
| 8,298 | 497 | 2.60 | 1.12 | 77.14 | 19.14 | .77 | .067 | 1.90 | 63.8 | Good coke, too much ash. |
| 7,163 | 461 | .42 | .43 | 84.34 | 14.81 | .83 | .086 | 1.89 | 55.1 | Improved by washing. |
| 4,256 | 229 | .27 | .78 | 82.97 | 15.98 | .82 | .087 | 1.89 | 42.5 | Improved by washing fine coal. |

*Coking tests*—Continued.

| Name of sample. | Coal. | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Condition. | Chemical composition. | | | | | Amount charged in oven. | Coking time. |
| | | Moisture. | Volatile matter. | Fixed carbon. | Ash. | Sulphur. | | |
| | | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Pounds.* | *Hours.* |
| West Virginia No. 6A | Raw | 1.76 | 21.48 | 71.92 | 4.84 | .55 | 13,000 | 90 |
| West Virginia No. 6B | do | 2.27 | 22.03 | 70.30 | 5.40 | .88 | 9,000 | 66 |
| West Virginia No. 6C | Raw, fine | 1.73 | 21.12 | 71.57 | 5.58 | .64 | 8,000 | 44 |
| West Virginia No. 7 | Raw | 3.85 | 20.55 | 70.58 | 5.02 | 1.36 | 8,000 | 44 |
| West Virginia No. 8 | do | 3.82 | 31.77 | 57.21 | 7.20 | .89 | 11,000 | 66 |
| West Virginia No. 9 | do | 3.81 | 31.08 | 57.04 | 8.07 | .83 | 10,000 | 40 |
| Do | Washed | 5.74 | 32.14 | 57.61 | 4.51 | .90 | 11,000 | 66 |
| West Virginia No. 10 | Raw | 1.61 | 18.39 | 74.65 | 5.37 | .61 | 11,000 | 68 |
| West Virginia No. 11 | do | 3.35 | 15.94 | 68.76 | 9.95 | .47 | 11,000 | 67 |
| West Virginia No. 12 | do | 1.53 | 18.23 | 74.08 | 6.16 | .97 | 11,000 | 68 |
| Do | Washed | 8.36 | 18.20 | 68.54 | 4.90 | 1.11 | 8,000 | 44 |

*Coking tests*—Continued.

| Coke. | | | | | | | | | | Remarks. |
|---|---|---|---|---|---|---|---|---|---|---|
| Amount produced. | Breeze and ash. | Chemical compositions. | | | | | | Specific gravity. | Per cent produced. | |
| | | Moisture. | Volatile matter. | Fixed carbon. | Ash. | Sulphur. | Phosphorus. | | | |
| *Lbs.* | *Lbs.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | *Per ct.* | | | |
| 8,303 | 381 | 2.83 | 2.17 | 89.84 | 5.16 | .69 | .008 | 1.77 | 63.9 | Gray, soft and light. |
| 6,399 | 382 | .36 | 1.34 | 93.04 | 5.24 | .76 | .007 | 1.87 | 71.1 | Better than A. |
| 5,849 | 273 | 1.89 | 1.83 | 88.80 | 7.48 | .69 | .006 | 1.79 | 73.1 | Dense, tough. |
| 5,119 | 287 | .67 | 1.23 | 90.40 | 7.70 | 1.03 | .0061 | 1.80 | 64.7 | Good coke. |
| 7,124 | 589 | 2.79 | .53 | 82.37 | 14.31 | .77 | .0116 | 1.92 | 64.7 | Hard, good color. |
| 6,084 | 314 | .29 | 2.41 | 88.15 | 9.15 | .82 | .0094 | 1.84 | 60.8 | Fair coke, heavy. |
| 6,803 | 78 | .27 | .62 | 91.73 | 7.38 | .77 | .012 | 1.87 | 61.8 | Very good coke. |
| 7,858 | 429 | 3.68 | .53 | 87.54 | 8.25 | .56 | .007 | 1.89 | 71.4 | Good coke. |
| | | | | | | | | | | Burned to ash. |
| 7,082 | 549 | .35 | .85 | 89.37 | 9.43 | .83 | .0077 | 1.95 | 64.4 | Fair coke. |
| 5,050 | 583 | 5.16 | .72 | 86.57 | 7.55 | 1.01 | .01 | 1.88 | 63.1 | Large pieces. |

# BRIQUETTING TESTS.

By JOSEPH HYDE PRATT.

## ACKNOWLEDGMENTS.

The writer was very ably assisted by Mr. A. A. Steel, who superintended most of the experiments made on the regular briquetting machines and conducted almost all of the laboratory work. Material assistance was also rendered by Mr. Charles H. Smith in suggesting ways and means and in actual experimental work about the plant. The success of the work is largely due to the Barrett Manufacturing Company, of St. Louis, and the Chatfield Manufacturing Company, of Carthage, Ohio, for supplies of pitch of various grades for binders; to Mr. C. C. Hawkins, chemist of the Chatfield Manufacturing Company, for suggestions and information regarding pitch, its analysis, etc.; and to Dr. Herman von Schrenk, of the Bureau of Forestry, for assistance in the rational analyses of pitch.

## BRIQUETTING PLANT.

The briquetting plant consisted of two machines of distinctly different types. One of the machines was furnished by William Johnson & Sons, of Leeds, England, and is designated in this report the English machine. This machine is one of the standard types for briquetting with stiff pitch, and consists of a double compression, vertical table press with closed molds and its accessories. It was operated by a 50-horsepower inclosed motor, which was furnished by the Westinghouse Electrical Manufacturing Company, of Pittsburg, Pa.

The second briquetting machine was furnished by the National Compressed Fuel Company, of Chicago, Ill., and consisted of a Chisholm, Boyd & White press, constructed on Belgian patterns and its accessories. This is ordinarily known as the eggette machine or press, but is designated throughout this report the American machine. It also was operated by a 50-horsepower motor, furnished by the Westinghouse Company.

### ENGLISH MACHINE.

In operating the English machine the coal is delivered from the washer or drier by an inclined conveyor, furnished by the Robins Conveying Belt Company, and received upon a slightly raised platform, holding about 10 tons, from which it can be conveniently shoveled through a hole in the same floor onto a coal-feeding worm. At the same time the pitch which is used as a binder, having been first reduced to a suitable size, is fed into the pitch cracker, which reduces it to a half-inch size and drops it to a smaller worm, and this in turn delivers it onto the coal in the larger worm. The smaller worm is driven by sprockets from the larger one, and the percentage of pitch that is added to the coal can be varied by changing the wheels.

While this arrangement gives good results when the same coal and binder are used, it is not sufficiently flexible to be adapted to the differences in behavior of the various coals and binders that were used in the experimental work; and therefore, for most of the experiments, the crushed coal and cracked binder were weighed out on Fairbanks scales and mixed by hand. The mixed coal and binder were then fed through the hole in the floor onto the larger worm.

After a slight mixing in the larger worm, the material enters an impact disintegrator, in which it is reduced to the desired degree of fineness. The speed of the disintegrator is varied according to the character of the coal. This is accomplished by running either one or both sides of the disintegrator and by tightening or slacking the belts. From the disintegrator the well-mixed, finely divided mass is elevated to a pug mill, about 5 feet in vertical height, in which the pitch or other stiff binder is softened by contact with live steam. As there is no attachment on this machine for introducing or utilizing superheated steam, the range of temperatures in which the binder could be softened was very limited. Then again, live steam is apt to be wet, and in such instances it would often be a detriment to the production of successful briquettes.

From the pug mill the plastic mixture falls by gravity to the feed box of the press, from which it is forced by a plunger into brass-lined molds in a vertical revolving table. After a half revolution of the table the mass in the mold is heavily pressed by a system of combined levers, the pressure being limited to 2 tons per square inch by the yielding of a stiff spring. This pressure, however, can be varied from a few pounds up to 2,000 pounds as a maximum.

After another quarter revolution the briquettes are forced, by means of a plunger, out of the molds on a slide or table set at a convenient height for stacking on trucks. In the experimental work at the plant the briquettes were taken from the slide or table by hand as soon as they were discharged from the press and stacked on a platform just outside the building. The briquettes, as they come from the machine, are rectangular, except for rounded corners, and weigh on the average 6.8 pounds each, the weight varying with the coal and binder used. The maximum capacity of this press is 6 tons of briquettes per hour.

This machine is adapted only to those binders which do not become plastic before reaching the pug mill, and in the experimental work the smallest quantity that could be made and at the same time give a satisfactory test to the coal and binder was 1 ton.

### AMERICAN MACHINE.

In operating the American plant the coal is received from the Robins conveyor at the level of the second floor in a bin of 3 tons capacity. From this it is spouted to a 500-pound bin on a Fairbanks platform scale, from which it passes into the boot of an elevator that hoists it up and drops it into a 15½ cubic foot measuring box. The coal is then dumped into a Buffalo mixer fitted with steam jackets for warming the coal and the binder. The binder is melted in a small steam jacketed tank entirely distinct from the main part of the machine, from which it is dipped by hand in the desired proportion and poured into the mixer. After a thorough mixing the mass is dropped into the feed slides of the press. This tangential press consists of two pairs of narrow-faced rolls, in the tores of which are ovoid cups for receiving the mixture to be pressed. As the rolls revolve the excess material is squeezed out, the resulting pressure being dependent upon the viscosity of the mixture that is being compressed. The eggettes are delivered from the press on a short rubber belt, and they weigh on an average about 0.3 of a pound each.

The capacity of the press is fully 5 tons of eggettes per hour, but the output of the mixing device was much too small for operating the press at its full capacity. As arranged, this machine was adapted for the use of soft binders only, and it was necessary to grind the coal at the washery. For experimental work, however, it had one advantage over the larger English press, inasmuch as it permitted the testing of mixtures in lots as small as 15 pounds. In making these small tests a 35-gallon camp kettle was employed for heating the mixture. It was also possible to test on the American machine the mixture used in the English machine by crushing up some of the hot briquettes as soon as they were received from the molds and transferring the material at once to the feed box of the American machine.

### BINDERS.

The materials that have been tried as binders in the laboratory experiments and on the two briquetting machines include the following: Pitch of various grades; creosote; asphalt, hard and soft, crude and refined; asphaltic pitch; petroleum, both of paraffin and asphalt bases; molasses; lime, and clay.

### PITCH.

Pitch is the residue left from the distillation of tar, which is produced (1) from by-product coke ovens; (2) from illuminating-gas plants; *(3) from producer-gas* plants; (4) from pintsch gas tar; (5) from water gas plants, and (6) from petroleum gas tar. Pitches made from the first three varieties of tar were tested, and each gave practically as good satisfaction as the other when approximately of the same composition. Petroleum gas tar gave the best results.

There was considerable variation noticed in the pitches that were submitted to the plant for the experimental work, and at the beginning it was difficult to obtain the grade that was desired. The first pitch received was from the St. Louis works of the Barrett Manufacturing Company. The pitch was too hard and would not soften at a sufficiently low temperature to be used satisfactorily in either the English or the American machine. It was subsequently proved that the pitch did not contain a sufficient quantity of the creosote oils to give it the required binding qualities. This pitch is designated *pitch A* throughout this report. The second lot of pitch was received from the Chatfield Manufacturing Company, of Carthage, Ohio. It was harder than pitch A, indicating that a still greater per cent of the creosote or adhesive oils had been driven off. Although these pitches would make briquettes, it was necessary to use from 13 to 18 per cent of them, while of a good, satisfactory pitch it required only from 6 to 9 per cent with the same coal. This pitch is designated throughout the report *pitch X.*

In order to determine the quality of pitch that would be satisfactory, some of the best tar from the producer-gas plant was heated until all of the water was expelled. The boiling was continued until, when the residue was dropped into water having a temperature of 55° F., it became brittle. After bucking down, this pitch stuck together on standing one week in the laboratory. It is known throughout this report as *pitch Z.* A long series of tests was then made with this pitch upon various coals, and it was found that 6 per cent of it made better briquettes than 13 per cent of *pitch X.* In order to further demonstrate the necessity for pitches of the right quality to contain a higher percentage of the creosote or adhesive oils than pitches A or X, experiments were made with hard pitch A and water-free producer-gas tar. These were melted together in various proportions until a product was obtained which, when dropped into water at 55° F., would become brittle. A sufficient quantity of this product was made to briquette 2 tons of coal, and it was found to give the results desired, showing that it is necessary for a pitch to contain a certain percentage of these creosotes or adhesive oils. This pitch will soften when held in the mouth; when it is first bitten it cracks and crumbles like spruce gum, but almost immediately becomes plastic and can be chewed like ordinary gum. This pitch is designated throughout this report *pitch Y.*

Samples of this pitch were sent to the Barrett Manufacturing Company and the Chatfield Manufacturing Company, and pitch of the right quality was obtained in quantity. The pitches that were received from the Barrett Manufacturing Company are designated in this report *pitches B*, *C*, and *D*. Those received from the Chatfield Manufacturing Company are known as *pitches E*, *F*, and *G*.

Another pitch that was furnished by the Barrett Manufacturing Company was obtained as a by-product from the manufacture of gas from heavy petroleum. When hot, this pitch has a marked odor of kerosene, but it would not mix with Kansas crude oil. It is lustrous and works well on a cool day, but is a little too soft in ordinary weather for experimenting on the English machine. This pitch is designated *pitch H* throughout this report.

Analyses have been made of the pitches A, B, C, D, E. F, G, H, and X by Mr. E. E. Somermeier, in charge of the chemical laboratory of the testing plant, with the following results:

*Proximate analyses of pitches.*

| Pitch. | Laboratory No. | Moisture. | Volatile matter. | Fixed carbon. | Ash. | Sulphur. | Total. |
|---|---|---|---|---|---|---|---|
| A | 1161 | 0.47 | 47.93 | 50.79 | 0.81 | [a] 0.80 | 100.80 |
| B | 1311 | 1.14 | 49.66 | 47.88 | 1.32 | .70 | 100.70 |
| C | 1391 | .88 | 62.75 | 35.84 | .53 | ........ | 100.00 |
| D | 1465 | 1.45 | 54.05 | 43.91 | .59 | ........ | 100.00 |
| E | 1464 | 1.02 | 54.11 | 44.04 | .83 | ........ | 100.00 |
| F | 1457 | .57 | 52.98 | 45.31 | 1.14 | ........ | 100.00 |
| G | 1453 | .60 | 52.53 | 45.62 | 1.25 | ........ | 100.00 |
| H | 1555 | 1.04 | 61.44 | 36.72 | .80 | ........ | 100.00 |
| X | 1125 | .33 | 59.07 | 39.44 | 1.16 | .88 | 101.38 |

[a] Approximately determined.

*Ultimate analyses of pitches.*

| Pitch. | Laboratory No. | Hydrogen. | Carbon. | Nitrogen. | Oxygen. | Sulphur. | Calorific value. |
|---|---|---|---|---|---|---|---|
| A | 1161 | ........ | ........ | ........ | ........ | 0.80 | ........ |
| B | 1311 | 3.97 | 90.89 | 1.05 | 2.07 | ........ | 8,782 |
| C | 1391 | 4.72 | 91.16 | 1.16 | 1.85 | .58 | 8,937 |
| D | 1465 | 4.28 | 91.57 | 1.10 | ........ | ........ | ........ |
| E | 1464 | 4.22 | 91.30 | 1.00 | ........ | ........ | ........ |
| F | 1457 | 4.16 | 91.50 | 1.00 | ........ | ........ | ........ |
| G | 1453 | 4.06 | 90.82 | 1.01 | ........ | ........ | ........ |
| X | 1125 | 4.56 | 90..34 | .99 | 2.07 | .88 | 8,847 |

Although no definite conclusions can be drawn from the results of the analyses given above, it is apparent that for pitches distilled from *the same tar, under* the same given conditions, those with the highest percentage of volatile matter give the best results in briquetting. There may be, however, a wide variation in the percentage of the volatile matter in two pitches made from different tars and distilled under different conditions and yet the pitches may have approximately the same quality for binding purposes.

In order to obtain something more tangible regarding the composition of the pitches in respect to their binding qualities and value in briquetting, a series of tests was made by distilling off thin volatile products. For this purpose a small copper still was constructed and the distillations were made in the laboratory of Dr. Herman von Schrenk. In carrying out these experiments, 200 grams of the pitch were placed in the still, and this was heated gradually until there was nothing left but a coke residue. In every instance at a temperature of 400° all of the volatile matter had been driven off and the residue was hard coke. The results of these distillations are given in the table below:

*Distillation of pitches.*

| Pitch. | Laboratory No. | Quantity of pitch used. | Temperature at which first drop came over. | Temperature at which the last of the oil was observed to come off. | Percentage of oil. | Final temperature to which still was raised. | Residue. |
|---|---|---|---|---|---|---|---|
| | | *Grams.* | ° | ° | | ° | |
| A | 1161 | 200 | 380 | 385 | 1.36 | 400 | Hard coke. |
| B | 1311 | 200 | 351 | ........ | 7.38 | 400 | Do. |
| D | 1465 | 200 | 332 | 375 | 13.01 | 400 | Do. |
| E | 1464 | 200 | 350 | 370 | 1.91 | 400 | Do. |
| G | 1453 | 200 | 324 | 382 | .96 | 400 | Do. |
| X | 1125 | 200 | 391 | ........ | .71 | 400 | Do. |

The above experiments show that from 7½ to 14 per cent of these oils is necessary in the pitches in order to give them the binding qualities that are desired in briquetting. With a higher percentage of oils the pitches become too soft and can not be used to so good advantage. Then again, the oils that are driven off from 315° and under are the creosote oils. Creosote was used to mix with some of the hard pitches for binding purposes and gave satisfactory results.

## ROSIN.

The cheaper grades of rosin can be used for binding purposes to as good advantage as the refined material, and, as far as could be judged from the work done in the laboratory, the crude, black rosin will give as good satisfaction as the partially refined. Rosin in some cases can be used to advantage for hardening purposes as well as for its binding qualities. It has been used with lime and pitch, and good briquettes were made from this combination. In using rosin and pitch as a binder it was found that a smaller percentage of these two together was required to give either as good or a better briquette than when pitch was used alone. In burning there is little or no odor from the rosin, although there is some tendency to smoke.

### ASPHALT.

The asphalt used in the experimental work consisted of a California refined asphaltum of grade B, which is designated throughout this report *asphalt B1.* Another asphalt product used is known as "kopak" No. 30, manufactured by the Raven Mining Company, of Texas, and is of a rubbery consistency. This is known as *asphalt B2.* An asphaltic pitch manufactured by the Standard Oil Company at one of its Texas plants also was used, and it is known as *asphalt B3.* Crude asphalt from Indian Territory is known as *asphalt B4.* This asphalt is altogether too hard, approaching close to a coke, and containing but a very small percentage of volatile matter.

A soft asphalt was received from the Gulf Refining Company of Port Arthur, Tex., and is designated throughout this report *asphalt B5.* An analysis of this asphalt by Mr. E. E. Somermeier was as follows:

*Analysis of soft asphalt from the Gulf Refining Company.*

| | Per cent. |
|---|---|
| Moisture | 0.00 |
| Volatile matter | 80.75 |
| Fixed carbon | 19.25 |
| Ash | 0.00 |
| Total | 100.00 |

Another soft asphalt was furnished by Mr. John McNeil from Casper, Wyo., and is known throughout this report as *asphalt B6.* It is a soft, tough asphalt, which on cooling to about 40° became brittle. An analysis by Mr. E. E. Somermeier gave the following results:

*Analysis of asphalt from Casper, Wyo.*

| | Per cent. |
|---|---|
| Moisture | 0.48 |
| Volatile matter | 78.77 |
| Fixed carbon | 20.75 |
| Ash | 0.00 |
| Total | 100.00 |

### PETROLEUM.

Petroleum, of both paraffin and asphaltum bases, can sometimes be used to advantage in briquetting. On account of lack of time but little experimenting was done with the petroleums. Those that have been used were crude oil from Spindletop, Tex., which was produced by the Gulf Refining Company, and a petroleum from the Beaumont district, having a specific gravity of 15, flash test 505, and fire test 570, which was produced by the Great Southern Refining Company. These are designated *P1* and *P2*, respectively, throughout this report.

The only other oil that was tested in briquetting was a Kansas crude oil, which is designated throughout this report *P3*.

### MOLASSES.

A few experiments were made with molasses in order to determine the possibility of utilizing waste products from sugar and beet refineries in briquetting lignite and lignitic coals that occur in the vicinity of sugar refineries. Although no positive results have been obtained with the use of this binder, the work has shown the probability that such waste products can be utilized in connection with certain other materials to give a satisfactory binder.

### LIME.

Lime has been used in combination both with rosin and with molasses, and some encouraging results were obtained. One of the main objections to using more than a very small amount of lime is the fact that it increases the percentage of ash in the fuel without raising its heating efficiency. On the other hand, however, it is one of the cheapest of binders, and in most cases is convenient to the coal districts.

### CLAY.

A few experiments were made with clay in combination with other compounds in an attempt to obtain a suitable binder for lignitic coals, but without success.

## LABORATORY EXPERIMENTS.

In testing the value of the various binders for briquetting coal, a series of experiments was made with each in the laboratory by briquetting the mixture in a hand press. The mixture of coal and binder was heated in a clay assay crucible over a Bunsen burner. In most cases they were heated only to about 212° F., but where the binder was softened with difficulty the temperature was increased.

### PITCHES.

Special experiments were made with the different pitches, A, B, C, etc., and where there was considerable variation in their composition a more nearly complete series of experiments was made. When, however, the pitches were similar, as B, C, and D, the series of experments made with one was significant of what could be done with the others.

*Pitch A.*—A series of experiments was tried in the laboratory with pitch A in order to determine its binding qualities. It was found that the best practical results were obtained with 9 per cent of this hard pitch on an easy briquetting coal, but with the other coals the percentage would be very much higher.

*Pitch B.*—This pitch was tested with West Virginia coking coal. With 3 per cent of pitch B a fair briquette was obtained. With 4 per cent the resulting briquette was a little better. With 5 per cent, however, a good briquette was obtained, which had the desired lustrous fracture. With Arkansas coal it required 6 per cent of this pitch to give the desired results; and as this coal represented more nearly the character of the coals to be briquetted, this was accepted as the percentage required of this and the similar pitches, C and D, to make a good briquette.

*Pitches C and D.*—These, which are very similar to E, were used from the formulas derived for B, and in nearly all cases the runs made on the English and American machines were satisfactory.

*Pitch E.*—This pitch, which melts at 180° F., is a trifle too hard for satisfactory briquetting on the English machine, and is not equal to pitch B, C, or D.

*Pitch G.*—This pitch, which melts at 196° F., is too hard (similar to A and X) to be used alone in briquetting.

*Pitch X.*—This pitch, which contained only 0.71 per cent of volatile matter, was tested with a number of coals and it required from 13 to 20 per cent to make a briquette that was at all coherent.

*Pitch Z.*—The producer-gas pitch Z was tested with Arkansas coal, 6.7 per cent of the pitch being used. While hot both of these briquettes were soft, but on standing one hour they became very hard and had a glossy or lustrous fracture. When tested in the muffle they burned without disintegration.

A series of experiments was made which indicated that there is no advantage in using lime with pitch in an attempt to reduce the percentage of pitch that it is necessary to use. With, however, the introduction of some third reagent which would react with the lime it is possible that a binder could be produced which would be cheaper than pitch alone and be as satisfactory.

## ASPHALT.

In experimenting with the asphalts, they were used alone as a binder and also in combination with other compounds. There were a number of satisfactory results obtained in the laboratory work, but only a very few of these could be tried on the English or American machines on account of their construction.

*California asphalt, B1.*—This asphalt is brittle at ordinary temperatures and can be readily disintegrated. At 212° F. it softens slightly, and it is soft and very sticky after melting. This asphalt mixes readily with the Spindletop (Tex.) petroleum, P1, and the residual product of Beaumont petroleum, P2, in all proportions, and can therefore be thinned to any consistency. A series of experiments was made with this asphalt, which indicates that to obtain the best results it would be necessary to heat the mixture with superheated steam or to add crude oil.

*Asphalt product (kopak), B2.*—This Texas asphaltic compound is of a rubbery consistency at ordinary temperatures, and at 212° F. only softened slightly without becoming sticky. It melted at a rather high temperature, was still rubbery, but very adhesive. Alone it can not be disintegrated readily, but in a cold temperature it can with some care be crushed. It is not affected by the crude Spindletop (Tex.) oil, P1, or the residual petroleum from the Beaumont district, P2. With the warm asphaltic pitch it mixes readily, and but little of this pitch is needed to make a soft and sticky mass. With ordinary coal-tar pitches this asphaltic compound will not mix at all. With rosin the rubbery asphalt mixes readily, and the resulting product can be cracked easily, and it is very strongly tenacious and somewhat elastic. This asphalt has the qualities that should make it a desirable binder, and it may be found useful in briquetting lignites.

*Asphaltic pitch, B3.*—This asphaltic pitch, made by the Standard Oil Company at one of its Texas plants, was found to have excellent binding qualities and to be of the right consistency to work easily. It makes a tough and elastic briquette, but apparently it does not give the degree of hardness that can be obtained from the pitches made from coal tar.

A series of experiments was made with this pitch in combination with rosin. The briquettes made from 4 per cent asphaltic pitch and 2 per cent rosin and from 3 per cent pitch and 3 per cent rosin were of equal value; but as the 3 to 3 briquette would probably be the cheaper, it has been adopted as the standard ratio for this asphaltic pitch and rosin.

*Indian Territory asphalt, B4.*—This asphalt does not soften at all in boiling water, even when tested as powder. It melts to a stiff, sticky liquid rather suddenly just below its ignition point, which is at a very high temperature. It is only slightly soluble in gasoline. At the boiling point there is apparently no reaction between this hard asphalt and crude Kansas petroleum, P3, or the Spindletop oil, P1, or the Beaumont residual petroleum, P2. Neither is there any apparent action between this asphalt and melted rosin. With creosote it softened materially, and when used in the ratio of 5 per cent of each with Arkansas coal a fair briquette was obtained. With Pintsch gas tar an apparent reaction took place, which caused the separation of some of the intrained water. This asphalt was tried alone with Arkansas coal, 4 per cent of asphalt being used, and even when heated dry at a high temperature the resulting briquette was poorly coherent. With 6 per cent of asphalt the briquette was little or no better; indicating that the asphalt alone has no binding properties, as was supposed from the very small amount of volatile matter that it contained.

*Wyoming asphalt, B6.*—This asphalt united readily with rosin, and in preparing a mixture for briquetting 2 parts of asphalt and 1 part of rosin were melted in a kettle and cooled over night. On a cool, frosty day this mass was very brittle, but became sticky with a rise in temperature. In using the mixture 1 to 2 parts of lime were added to prevent its becoming too soft while hot. The briquettes made from this mixture were all right in appearance, but in burning they crumbled easily, not standing up well in the fire.

In testing this asphalt alone it was necessary to heat the coal in an assay crucible, adding the fragments of asphalt and kneading them together with repeated heatings. Six per cent of this asphalt alone would make a strong, tough briquette with Wyoming lignite, which was slightly earthy in appearance. When tested in the fire, however, the briquette fell to pieces at once and completely.

### ROSIN.

In using rosin alone it was found to make hard, brittle briquettes, but they were not satisfactory, principally on account of their brittleness, which would prevent their standing rough treatment in transportation. The results of the tests made with rosin and pitch have been recorded under "Pitch." Rosin mixes with almost all the crude oils and their residual products which have been tested in the laboratory. With the rubbery asphalt, B2, it mixes readily, forming a still more rubbery mass. With the Indian Territory, B4, it does not unite at all. All of the oils and petroleums increased the toughness of the rosin, and the resulting masses varied from the consistency of wagon grease to sticky rosin. Five parts of rosin melted with 1 part of Kansas crude petroleum gave a tough mass that could just be cracked when cold, and this seemed to be the best ratio for these two compounds. With lime alone it was found that 15 per cent of lime and 85 per cent of rosin were required for a complete reaction. Rosin with unslacked lime sets after melting. If the slacked lime is used the action occurs at 212° F. and the mass is light gray. This cement softens and becomes sticky only at the melting point of lead. On cooling it is brittle, but stronger than rosin. With slacked lime rosin reacts and gives a satisfactory mixture up to 3 parts of rosin and 1 part of slacked lime. A combination of rosin and the residual Beaumont petroleum, *P2*, was made considerably tougher by the addition of lime, and for this relatively more lime is needed than when used with rosin alone. The best combination obtained was 6 parts of rosin to 3 parts of slacked lime to 2 parts of the petroleum, P2. With Kansas crude oil no positive results could be obtained. Six parts of rosin, 3 of lime, and 4 of the Kansas oil, P3, gave a greasy granular mass without any apparent binding properties. With 3 parts of oil instead of 4 a sticky mass was obtained, which, however, had no strength.

### PETROLEUM.

All the crude oils and petroleums can be stiffened to some extent by mixing with rosin or hard pitch. None of the petroleums and oils that have been tested, however, could be used alone in briquetting any of the coals. Most of them were too fluid and had little or no binding qualities.

### MOLASSES.

The molasses experimented with in the laboratory was that which can be bought in any market. In using the thick molasses it was found advantageous to dilute it with twice its weight of water, and the lime was slacked to a paste with twice its weight of water. When these two reagents were mixed together they had no apparent reaction until dried out. Using 1 part of water to 1 of molasses and 2 of lime, there was no apparent excess of either, and the mixture set at once to a rather hard cement, similar to plaster of Paris. A series of experiments was made with Arkansas coal, which showed that 5 parts of molasses to 1 of lime would give the best results.

### WAX TAILINGS.

A sample of wax tailings was received from the Standard Oil Company, which was tested with Arkansas coal. With 2 per cent of wax there was no coherence between this and the coal under moderate pressure. With 4 per cent, either hot or cold, an elastic briquette was obtained which was fairly clean, but yielded under pressure applied slowly. These briquettes were burned in a scorifier placed in a muffle, and they seemed to burn perfectly. With pressure a briquette with 4 per cent was obtained which had no greater strength than the other, but was very smooth and clean. With 5 per cent of the wax and with light pressure a briquette was obtained that was rather sticky, and there was undoubtedly an excess of wax, as was indicated by its sticking to the plunger.

The wax was also tried with some of the harder pitches. With 1 per cent of wax and 3 per cent of pitch X a briquette was obtained that had no coherence whatever. With 1 per cent of wax and 5 per cent of pitch the resultant briquette had no coherence, regardless of what pressure was used. With 2 per cent of wax and 4 per cent of pitch X a briquette was obtained which was similar to the one in which 13 per cent of pitch X alone was used. It was, however, not a satisfactory product.

### ACID SLUDGE.

Some acid sludge was received from the Gulf Refining Company, of Port Arthur, Tex. It was mixed with water, and the resultant solution was a weak acid which did not seriously attack iron. Part of this sludge was a thin, greasy mass and the remainder was of a rubbery, granular consistency. When these were heated together to the melting point they did not mix, and on cooling they separated into the same two parts. The stiffer portion of the mixture will mix with melted rosin, and the resultant mass resembles the rubbery petroleum, B2. On account of lack of time no further work was attempted with this acid sludge.

### LIME.

No experiments were made with lime alone, as the percentage required for making a briquette that would hold together was so high that it precluded entirely its use as a binder. The principal use of the lime was with rosin and molasses, and these experiments have been described under those heads.

### CLAY.

Clay alone was tried up to 8 per cent, but in no case was a briquette obtained that would hold together on drying, although the clay used was a superier potter's clay. In making the laboratory briquettes the clay was used only in connection with lignite, in order to determine its effect on the strengthening of the briquettes so that they would hold together in burning. With 2 per cent of clay there was little or no effect noticed in the resultant briquette when rosin and the petroleum, P2, were used. With 4 per cent of clay the quantity of the other two required was reduced from 6 per cent each to 5½ per cent each, and the briquettes that were made from this mixture held together somewhat better in the fire. On account, however, of the large increase in the percentage of ash caused by the clay no further tests were made with a higher percentage of clay.

## BRIQUETTES MANUFACTURED.

According to the results obtained in the laboratory, briquettes in quantity were made on either the English or the American machine from the following coals:

## LIST OF COALS BRIQUETTED.

(1) **Alabama** No. 1. Bituminous lump and nut coal from mine No. 8 of the Ivy Coal and Iron Company, Horse Creek, Ala.

(2) **Arkansas** No. 1. Bituminous lump and nut coal from mine No. 3 of the Central Coal and Coke Company, Huntington, Ark.

(3) **Arkansas** No. 2. Bituminous lump coal from mine No. 12 of the Central Coal and Coke Company, Bonanza, Ark.

(4) **Arkansas** No. 3. Bituminous lump and slack coal from mine No. 18 of the Western Coal and Mining Company, Jenny Lind, Ark.

(5) **Arkansas** No. 4. Semibituminous slack coal from several Arkansas mines, furnished by the Western Coal and Mining Company, St. Louis, Mo. The coal came principally from mines 1, 2, 3, and 4, near Denning, Ark.

(6) **Arkansas** No. 5. Semibituminous lump and slack coal from mine No. 4 of the Western Coal and Mining Company, Coal Hill, Ark.

(7) **Arkansas** No. 6. Bituminous slack coal from mine No. 18 of the Western Coal and Mining Company, Jenny Lind, Ark. This is from the same mine as Arkansas No. 3.

(8) **Colorado** No. 1. Run-of-mine black lignite from the Simpson mine of the Northern Coal and Coke Company, Lafayette, Colo.

(9) **Illinois** No. 1. Bituminous lump and nut coal from mine No. 1 of the Western Anthracite Coal and Coke Company near O'Fallon, Ill.

(10) **Illinois** No. 4. Bituminous lump coal from mine No. 3 of the Donk Brothers Coal and Coke Company, Troy, Ill.

(11) **Indiana** No. 1. Bituminous run-of-mine coal from the Mildred mine of the J. Woolley Coal Company, Mildred, Ind.

(12) **Indiana** No. 2. Bituminous run-of-mine coal from the Electric mine of the T. D. Scales Coal Company, Boonville, Ind.

(13) **Indian Territory** No. 2. Bituminous run-of-mine coal from mine No. 8 of the Rock Island Coal Company, Hartshorne, Ind. T.

(14) **Indian Territory** No. 3. Bituminous run-of-mine coal from mine No. 1 of D. Edwards & Son, Edwards, Ind. T.

(15) **Indian Territory** No. 6. Bituminous slack coal from mine of the Southwestern Development Company, Coalgate, Ind. T.

(16) **Iowa** No. 4. Bituminous lump coal from mine No. 3 of the Centerville Block Coal Company, Centerville, Iowa.

(17) **Kansas** No. 2. Bituminous lump, nut, and slack coal from mine No. 11 of the Western Coal and Mining Company, Yale, Kans.

(18) **Kentucky** No. 1. Bituminous run-of-mine coal from Straight Creek mine No. 2 of the National Coal and Iron Company, Straight Creek, Ky.

(19) **Kentucky** No. 2. Bituminous lump, nut, and pea coal from mine No. 11 of the St. Bernard Mining Company, Earlington, Ky.

(20) **Missouri** No. 1. Bituminous run-of-mine coal from New Home mine No. 1 of the New Home Coal Company, Sprague, Mo.

(21) **Montana** No. 1. Washed bituminous coal from mine near Red Lodge, Mont.

(22) **New Mexico** No. 1. Black lignite from Weaver mine of the American Fue Company, 3 miles north of Gallup, N. Mex.

(23) **New Mexico** No. 2. Black lignite slack from the Otero mine of the Caledonian Coal Company, 2 miles east of Gallup, N. Mex.

(24) **North Dakota** No. 1. Run-of mine brown lignite from Lehigh, N. Dak.

(25) **Pennsylvania** No. 3. Anthracite culm from Pennsylvania Coal Company, Scranton, Pa.

(26) West Virginia No. 6. Bituminous run-of-mine coal from mine of the New River Smokeless Coal Company, Rushrun, W. Va.

(27) Wyoming No. 1. Black lignite from mine of the Wyoming Coal and Mining Company, located at Monarch, Wyo.

The above coals have been tested with various binders on both the English and American machines, and from a ton up to 15 tons of briquettes have been made from each coal. In the following pages is given a description of the binders used, of the character of the briquettes as they were received from the machine and after they were exposed to the atmosphere, and of their behavior when burned. These descriptions are taken up alphabetically by States.

### ALABAMA.

*Alabama No. 1.*—This is a coking coal which can be very readily manufactured into a briquette with hard pitch, and which under extreme pressure can be briquetted without the addition of any binder. There were 4.5 tons of this coal briquetted on the English machine with 7 per cent of the hard pitch A. The briquettes were strong and quite satisfactory, except for porosity due to a lack of sufficient pressure. These briquettes stood up well in the fire,[a] but on long exposure to the weather became somewhat disintegrated. The weight of the briquettes averaged 5½ pounds each.

### ARKANSAS.

All of the Arkansas coals tested are similar in their properties. They are higher in fixed carbon than the ordinary bituminous coal and are often called semianthracite coals. The coals from Jenny Lind and Denning show more of this character than the coals from the other sections of Arkansas.

*Arkansas No. 1.*—This coal was tested only with hard pitch A, as this was the only pitch available at the time the coal was briquetted. Six tons of coal were mixed with 9.25 per cent of this pitch and made into briquettes on the English machine.[a] The briquettes were compact and well pressed, but were too friable for handling, showing that there was not sufficient pitch. They were brownish in color and very dirty. Judging from the results obtained in working with other Arkansas coals, this coal will require more than the usual amount of pitch.

*Arkansas No. 2.*—This coal was tested with the very hard pitch X, which at the time was the only pitch available. These briquettes were made in the English machine, which was set to give approximately 11 per cent pitch. There were 6 tons of crushed lump coal run through the machine, but the resultant briquettes were very pitchy and wrinkled and probably contained nearer 15 per cent than 11 per cent pitch.[a] On account of the difficulty of setting the machine to feed accurate percentages of pitch to the coal, the experiments were conducted for the most part after this by weighing out the determined quantities of coal and pitch by hand. Pitch X, on account of its hardness, set too quickly in the molds, so that the briquettes were insufficiently pressed and apt to be porous.

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[a] For steam test see p. 80.

*Arkansas No. 3.*—This coal was briquetted without running it through the drier with 9.5 per cent of the hard pitch X, which was found to be plenty for this coal. The coal was slightly moist, which made it in just the right condition to pass readily through the machine without clogging the elevator. The coal was very friable and was reduced to almost a flour in the disintegrator. The briquettes had very smooth, polished surfaces, having received the greatest pressure that could be obtained from the machine. Some of the briquettes, as they were received from the machine, were cracked perpendicularly to the pressure, which may have been due to an excess of pressure. These briquettes weighed on an average 6.8 pounds each.

This coal was also tested with the somewhat softer pitch A. One ton of the coal was mixed with 8.7 per cent of pitch. A large excess of steam was introduced into the pug mill so as to soften the pitch as much as possible. The briquettes were smooth and easily handled, but somewhat pitchy, showing that there was an excess of pitch. Under the same conditions of heat and pressure, 7.5 per cent of pitch A would have been sufficient, and if a good pitch were used a still smaller amount would make a good commercial briquette. These briquettes ignited readily and did not disintegrate in the fire until they were consumed.[a]

*Arkansas No. 4.*—This consisted of slack coal, which was soaking wet when received, and it had to be passed through the drier before it could be used. When dried, it was so brittle that the disintegrators of the English machine reduced it to flour, so that it choked up the elevator and other parts of the machine. The first run made of this dried coal contained 12 per cent of hard pitch X, and although the briquettes when first received from the machine were very compact and smooth, they were incoherent and unsatisfactory and on exposure to the weather very soon began to disintegrate more or less and became friable. The next test on this very finely divided coal was with pitch A. One ton was mixed with 10 per cent of pitch. The press was run faster, not only to make the pressure greater, but also to give the hard pitch less time to set in the machine. At the same time the disintegrator was run slower in order not to reduce the coal to so fine a condition. The briquettes were very pitchy, but owing to the excess of water from the condensation, many cracked when the pressure was relieved. Another ton of coal with 8 per cent of pitch A contained too much binder, and another ton was made with 6 per cent. These latter briquettes were compact and well pressed, but did not contain enough binder. Seven per cent of pitch A gave a better result than either 6 or 8.

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[a] For steam test see p. 80.

Another run of this coal was made with 3 per cent rosin and 2 per cent of pitch A as a binder. The briquettes were very clean and sharp in outline, but were somewhat brittle and had a tendency to break into large fragments. They were, however, physically better than those with 6 or 8 per cent of pitch alone. In burning, they held together very well, but smoked more than those with pitch alone, but no odor of rosin was given off. This same mixture was tested on the American machine, which gave clean, polished briquettes, which were stronger than those made on the English machine, indicating that rosin works to better advantage in the smaller and more rounded briquettes.

A ton of this coal was briquetted by using 3 per cent of rosin and 3 per cent of pitch A. This mixture made excellent briquettes, which were usually clean and with sharp edges. It was also tried on the American machine and gave exceedingly smooth and lustrous eggettes, which were stronger than the large briquettes made on the English machine. The eggettes were tested in a cook stove, where they burned very satisfactorily without any odor, but gave off considerably more smoke than either coal or briquettes made from pitch alone as a binder.

This coal was further tested with the softer pitch B, which was one of the better pitches for briquetting. One ton was first made with 6 per cent of pitch, and the briquettes were clean, sharp, and apparently considerably stronger than the original lump coal. These were the first commercial briquettes made at the plant. They were capable of standing very rough handling. In weight they averaged 7 pounds each, and their specific gravity was 1.17. On the American machine this same mixture gave polished eggettes, but they were not so strong as those made on the English machine. Pitch B was then tried with rosin, the proportions being 95 per cent of coal, 3 per cent rosin, and 2 per cent pitch. On account of the condensation of steam in the pug mill, there was considerable excess of water in the briquettes, which, however, were smooth and sharp when received from the press. The excess of moisture caused them to crack badly, and they were difficult to handle while fresh. After cooling, they were still soft, and they were noticeably inferior to the briquettes made with 3 per cent of rosin and 2 per cent of pitch A. They weighed 7.2 pounds each. This mixture also was tested on the American machine, which made harder eggettes than the corresponding briquettes of the English machine. This is a further confirmation of the supposition that the smaller and rounder form is better adapted for the rosin binder and will result in a harder and tougher briquette.

The manufacture of briquettes from Arkansas No. 4 coal and 6 per cent of pitch B as a binder is commercially feasible. Six tons of this coal were briquetted with 6 per cent of pitch B, with which to make a boiler test.[a] A portion of these briquettes were made by running but one side of the disintegrator of the English machine, and as they were received from the machine they were smooth and uniform but very soft, and on breaking them open there were spots that contained an excess of pitch, indicating that the materials had not been thoroughly mixed or that in passing from the pug mill to the press there had been an opportunity for the pitch to flow together. On cooling, however, these briquettes became very hard and tough. Another ton was made, running both sides of the disintegrator, which made smooth, uniform briquettes, but it was almost impossible to handle them as they came from the machine. Therefore, only one side of the disintegrator was used, but the materials were run through the pug mill more rapidly and better results were obtained. On becoming thoroughly cold, all of these briquettes were excellent in every respect. When these mixtures were tried on the American machine, the fine mass obtained by running both sides of the disintegrator made better eggettes than the corresponding briquettes of the English machine, while the coarser material did not make as good eggettes as briquettes. The weight of the briquettes obtained in running both sides of the disintegrator was 6.92 pounds each, while the coarser briquettes weighed 6.56 pounds each. The specific gravity of these briquettes was 1.17 and their crushing strength was 17,500 pounds per square inch.

*Arkansas No. 5.*—On account of the success obtained with these Arkansas coals and the softer pitches, it was decided to make a sufficient quantity of briquettes for a locomotive test on the Missouri Pacific Railroad. For this purpose 17 tons of Arkansas No. 5 coal were briquetted with 6 per cent of pitch C, which was a trifle harder than pitch B. Excellent briquettes were obtained, which were readily handled while warm, and on cooling stood a good deal of very rough handling. They did not break so readily as the original lumps of coal. The briquettes were tested on a locomotive of the Missouri Pacific Railroad, and for comparison similar runs were made on an Illinois run-of-mine coal such as is commonly used on these locomotives. Three trips were made from St. Louis to Washington, Mo., and return, making a total run of 324 miles for each fuel. The results of these three runs are given in the tables following.

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[a] For result of test see p. 80.

*Locomotive test of briquettes of Arkansas No. 5 coal.*

| Date. | Briquettes consumed. | Ash removed. | Front-end cinders removed. | Total coal consumed. | Total ash removed. | Per cent of ash. | Miles to ton. |
|---|---|---|---|---|---|---|---|
| | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* | *Pounds.* | | |
| November 28 | 9,720 | 1,250 | 194 | 29,594 | 4,615 | 15 | 21.89 |
| November 29 | 10,049 | 1,535 | 182 | | | | |
| November 30 | 9,825 | 1,226 | 228 | | | | |

*Locomotive test of run-of-mine Illinois coal.*

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| December 1 | 11,265 | 1,980 | 130 | 34,015 | 5,695 | 16.4 | 19.05 |
| December 2 | 11,050 | 1,558 | 112 | | | | |
| December 3 | 11,700 | 1,800 | 115 | | | | |

A further test of Arkansas No. 5 coal was made by mixing pitches C and A up to one-fourth of the soft pitch C and three-fourths of the hard pitch A, and a briquette of good quality was obtained, but it was not equal to the briquette made with 6 per cent of pitch C. With more than three-fourths of hard pitch, it was necessary to increase the percentage slightly, to use more steam for heating and a greater pressure, and the resultant briquettes were considerably inferior and more crumbly.

*Arkansas No. 6.*—This coal was briquetted with 8 per cent of pitch A and made a briquette of fair quality. These briquettes were made in order to test the coking value of the mixture, and this work is described on pages 122 and 168.

### COLORADO.

*Colorado No. 1.*—The only coal that was tested from Colorado is a black lignite, which was first used undried with 10 per cent of pitch D. As these briquettes were received from the machine, they were hard to handle, especially when there was an excess of steam. The pitch did not seem to adhere to the grains of lignite. On cooling, however, they became hard, black, and lustrous, but were too brittle. The adhesion of the pitch to the grains of lignite was not very great. The briquettes seemed to be too pitchy, and if the adhesion between the pitch and lignite were greater 8 per cent would undoubtedly be sufficient to give the desired results. Although there was no steam test made with these briquettes, they were burned under one of the boilers and proved quite satisfactory. They weighed 5.98 pounds each. Some of the same mixture was tested on the American machine, but the *eggettes* were much inferior to the briquettes.

### ILLINOIS.

*Illinois No. 1.*—Briquettes from this coal were first made with the hard pitch X as a binder. One and a quarter tons of this coal were briquetted with 16 per cent of pitch X, but on account of the softness and roughness of the coal the briquettes were brown and very soft, resembling pressed damp loam. They were, however, compact and well pressed.

*Illinois No. 4.*—Two tons of this coal were briquetted with 6 per cent of pitch D. The coal was not ground very fine, there being but one side of the disintegrator used. There was a great deal of water in the steam, with the result that the briquettes were full of the usual perpendicular cracks. Very great pressure was used, and some of the briquettes were larger than the normal size, which was due to the compression of the spring of the machine. The average weight of these briquettes was 6.95 pounds. The briquettes varied considerably in texture, some being hard and firm while others were crumbly. On cooling, the briquettes were hard and firm, but would not stand very much rough handling. The specific gravity of these briquettes was 1.16, and their crushing strength was only 5,100 pounds to the square inch. Some of these mixtures were also tried on the American machine, and the eggettes were better than the briquettes made on the English machine. They were lustrous and had a deep black color.

One ton of this coal was briquetted with 8 per cent of pitch D, using but one side of the disintegrator and with the maximum pressure. Splendid briquettes were obtained with this combination, but they were gray in color. They weighed 6.43 pounds each and had a specific gravity of 1.11. The specific gravity of the coal is 1.34. The crushing strength of these briquettes was 9,327 pounds, as compared with 5,100 when 6 per cent pitch D was used as a binder. The eggettes made of this mixture on the American machine were not so strong as the briquettes.

The next test made with Illinois No. 4 consisted of 2 tons of the coal briquetted with 8 per cent of pitch E. The average weight of the briquettes was only 6.15 pounds. Only one side of the disintegrator was used, and as the coal was hard the briquettes were coarse and contained many noticeably large pieces of coal. The eggettes made from this mixture, on the other hand, were much better in quality and quite lustrous, but of a decidedly brownish color. The specific gravity of these briquettes was 1.13 and their crushing strength 7,265 pounds to the square inch. To test the effect of finer grinding, another ton of this coal was briquetted with 8 per cent of pitch E, using both sides of the disintegrator. In these briquettes there were a few grains of coal as big as a kernel of corn; but for the most part the material was too fine for the percentage of pitch, and the resultant briquettes were crumbly, although well pressed, black, and lustrous. They weighed on an average 6.5 pounds each and had a specific gravity of 1.03. Their crushing strength was only 6,000 pounds to the square inch. There was little difference in the properties of the eggettes made of this mixture and the briquettes. Another ton of this coal was briquetted with 9 per cent of pitch E, using both sides of the disintegrator with a maximum pressure, with the result that a briquette of much better quality was obtained. They could, however, have been still further improved by the addition of another one-half per cent of pitch. The weight of these briquettes was 6.43 pounds each and their specific gravity 1.08. Their crushing strength was 9,125 pounds to the square inch. The eggettes made from this mixture were equal in strength and quality to the briquettes and were much more lustrous and bluish black in color.

Another test was made with Illinois No. 4 coal and pitch H, which is a by-product from the manufacture of gas from heavy petroleum. While this pitch would be rather soft to work on the English machine on a hot day, it is readily cracked and mixed on a cold day. One ton of the coal was briquetted with 8 per cent of pitch H, and the resultant briquettes were very strong, clean, and satisfactory, being very much better than any of the other briquettes made with this coal. They were readily handled when taken from the machine, and when cold were very hard and tough, and were capable of standing a great deal of rough handling. Their average weight was 6.81 pounds each and their specific gravity was 1.11. Their crushing strength was 12,810 pounds to the square inch, this being the greatest of all the briquettes made from Illinois coal.

The last test made with this coal was with a soft asphalt received from the Gulf Refining Company. In freezing weather this asphalt became hard enough to crack and thus could be used on the English machine. On account of the cold weather, there was considerable water in the steam, which produced cracks. Because of this fact and the low melting point and thinness of the asphalt while hot, the briquettes were difficult to handle as they came from the machine. They were, however, well pressed and on cooling became tough and would stand much handling. A better result can probably be obtained by using a small percentage of rosin with the asphalt.

## INDIANA.

Two coals from this State were tested more for comparative purposes than with regard to their commercial possibilities.

*Indiana No. 1.*—Four tons of this coal were briquetted with 7 per cent of pitch D.[a] The coal was first washed and then dried. The briquettes were brown in color, readily handled as they came from the machine, and when cold stood rough handling. The average weight of the briquettes was 6.8 pounds each. The eggettes of this mixture, made on the American machine, were better and stronger than the briquettes.

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[a] For results of steam test see p. 80.

*Indiana No. 2.*—One ton of this coal was briquetted with 7 per cent of pitch H, and the resultant briquettes were satisfactory in every way. They were brown in color and resembled those made of Indiana No. 1 coal, but were much stronger, due to the better quality of pitch. They weighed on the average 6.42 pounds each.

### INDIAN TERRITORY.

*Indian Territory No. 2.*—This coal was tested with the hard pitch X, using first 8.5 per cent of pitch; but the briquette was not strong and crumbled very readily. With 12 per cent of pitch the briquettes were not much better, and with 13.4 the briquettes looked well as they came from the press, but were not strong and began to lose their corners soon after being exposed to the weather. Judging from the runs made, this coal will require more than the usual pressure. The briquettes containing 8½ per cent of pitch weighed only 5.08 pounds each. Those with 13.4 per cent of pitch weighed 6.15 pounds each.

*Indian Territory No. 3.*—This coal, which is hard and glossy, was tested on the English machine with 8 per cent of pitch A. This made fairly good briquettes, which were clean and hard, although somewhat porous, but they were not so strong as was desired. This defect was due in part to lack of sufficient heat in the pug mill to thoroughly soften the pitch. The briquettes as received from the machine weighed 6.56 pounds each. With 7.2 per cent of pitch A, this coal made smooth and fairly hard briquettes, which weighed 6.71 pounds each. With 6 per cent of pitch A, briquettes were made of this coal that were smooth and well pressed, but soft. These averaged in weight 6.8 pounds each. If this pitch could have been heated in the English machine to a sufficiently high temperature to cause it to soften and mix with the coal, it would undoubtedly have made a very good briquette; this is indicated by the fact that with a soft pitch, as pitch D or pitch H, this coal could be briquetted with 6 per cent of binder.

*Indian Territory No. 6.*—This sample consisted of a carload of very dirty slack, which contained considerable fire clay and some shale. Laboratory experiments had shown that 7 per cent of a binder of the quality of pitch D was the minimum that could be expected to give satisfactory results. Accordingly, one ton of this coal was mixed by hand with 7 per cent of pitch D, but the resultant briquettes were very crumbly and not at all satisfactory, although dense and well pressed. They weighed 7.35 pounds each. They were full of gray streaks on the surface, due to the fire clay, and on standing became coated with a heavy gray efflorescence, which was tested and proved to be calcium sulphate. Eggettes of this same mixture made on the American machine were practically the same in character and unusually dull grayish in appearance. The specific gravity of these briquettes was 1.205. When tested upon the compression machine these briquettes, which were very crumbly and weak and broke rather readily in the fingers, compressed much more than usual before breaking. Their crushing strength was 5,600 pounds per square inch. Another ton was tried with 9 per cent of pitch D, but the resulting briquettes were practically the same as with 7 per cent, although a little darker in color. They also showed similar streaks of fire clay on the surface and their fracture was decidedly earthy. They weighed on the average 7.5 pounds each. There seems to be but little adhesiveness between this pitch and the dirty coal, and to make a good briquette with this pitch it would be necessary to use probably as much as 15 per cent or more.

Satisfactory results with soft pitch not having been obtained, one ton of this slack coal was tried with 8 per cent of the harder pitch A. The briquettes were well pressed and blacker than those with 7 per cent of pitch D, but were not so strong. There was not the slightest tendency for the mixture to become stiff in the machine, which is a further indication of the destructive character of this coal on the properties of the pitch.

This coal was next tried by mixing together 93 per cent of coal, 6 per cent of rosin, and 1 per cent of quicklime. The lime was mixed in lumps from dust to three-fourths of an inch in diameter, which it was expected would be reduced by the disintegrator and slacked by the steam of the pug mill, so that the cement would not set too quickly. The disintegrator, however, had very little effect on the hard lime, and consequently there were many large unslaked pieces in the resulting briquettes, which caused them to fall to pieces. Some of this material that remained in the pug mill for some time gave a briquette that was hard while hot, but after cooling the briquette became earthy. These briquettes weighed on an average 7 pounds each. The eggettes made of this material were dense, but very crumbly.

In order to overcome the chances of unslaked lime going into the briquettes, the lime was slaked before using with one-half its weight of water, making a fine, dry powder. This was mixed with the coal to give the same proportions with the rosin as in the above test, and although the briquettes were smooth and well formed, having been thoroughly pressed, they were crumbly and not at all strong. They were grayish in color and similar in quality to those made with 7 per cent pitch D. They weighed on an average 7.5 pounds each. The eggettes of this same material were very crumbly and dull in luster.

Some of these lime-rosin briquettes were burned, and they were found to give less smoke than those with the rosin and no lime, and some less even than those with pitch alone.

Another test of this coal was made with rosin and Kansas crude oil. The experiments in the laboratory had shown that the best results could be obtained by using 6 per cent rosin and 1 per cent of the oil. In preparing this combination for the English machine, the oil was first mixed with five times its weight of fine coal, so as to make a nonsticky mass that would readily mix with the cracked rosin and coal in the usual manner. On account of the rosin becoming very fluid at 212° F., the hot briquettes as they were received from the machine had no strength, and cracked slightly under their own weight when piled only two high. They also cracked rather badly owing to the water from the wet steam. On cooling they became strong and were of good quality. Some of the mixture that was allowed to stand in the mold before pressing until cool enough to be plastic was very good as received from the machine, but it had not been sufficiently pressed. On standing, these briquettes became coated with a heavy brownish-gray efflorescence. The weight of these briquettes averaged 7.29 pounds each and their specific gravity was 1.25. The crushing strength of the briquettes was on the average 6,400 pounds per square inch. On account, however, of the lack of uniformity of the briquettes, the results of 8 tests varied from 4,300 to 9,240 pounds. The eggettes made from this same mixture were hard and compact, although of a dull color, and they had a good degree of strength as received from the machine. With an arrangement for cooling the mass after it has passed through the pug mill and before being fed to the press, this mixture would make briquettes which would be very good physically, but which would smoke rather badly.

Since the above tests showed that the best way to counteract the effect of the large amount of fire clay in this coal was to add some liquid binder which would soak into it and yet at the same time unite with the solid binder, another test was made with creosote as a liquid binder and pitch as the hard binder. For this purpose Indian Territory No. 6 coal was mixed with 8 per cent pitch G and 1¼ per cent of creosote, such as is used in preserving timber. The creosote and five times its weight of fine coal were first mixed together and then thoroughly mixed with the cracked pitch and coal. In operating the machine with this mixture, the amount of steam used for heating the mixture was gradually increased until there was an excess. This, however, had no effect upon the cooled briquettes, but it made the hotter ones more tender and a little harder to handle. The first briquettes of this mixture cracked a little, due to excess of water, but the remainder came from the machine strong enough to bear handling at once. The effect of the excess of steam was to insure more complete softening of the pitch and more uniform mixing of the clay. These briquettes were black and clean and only slightly streaked with the clay. When piled up they stuck together slightly, and were mottled a little from the creosote. They have, however, much less of the gray efflorescence than any of the Indian Territory No. 6 briquettes that have thus far been made, and are also the best that have been made, although still earthy. They are smooth and black, and although crumbling but little or not at all in handling, on breaking for firing they will crumble a great deal. The eggettes, on the other hand, made of this same mixture on the American machine are just as strong and would need no breaking in firing. The weight of these briquettes was 7.25 pounds each on an average. The specific gravity was 1.215, and their crushing strength was 6,500 pounds to the square inch.

Another test with the Indian Territory No. 6 coal was with the asphalt B5, which during the cool weather had become hard enough to crack, and thus could be readily used on the English machine. One ton of this coal and 8 per cent asphalt were used, which made briquettes that were soft and hard to handle while warm, especially when they contained moisture due to the wet steam. The better ones were compact, heavy, and tough, and although they had an earthy, oily appearance, did not crack when struck a severe blow. They would stand rough handling, but would be improved with a small percentage of rosin. These briquettes weighed 7.12 pounds each, and had a specific gravity of 1.22. Their crushing strength was very low, averaging 3,200 pounds per square inch. In making the compression tests the briquettes yielded a great deal before breaking and would be stronger under a quick load. The results varied from 1,800 to 5,100 pounds.

On obtaining a sample of pitch H it was decided to try the effect of this pitch on the Indian Territory No. 6 coal. The best results were obtained with this as a binder. One ton of coal was mixed with 8 per cent of pitch H, and with nearly dry steam and good pressure they made hard, compact briquettes. Although somewhat earthy, they would stand rough handling. Their average weight was 7.27 pounds each.

## IOWA.

*Iowa No. 4.*—One ton of this coal was briquetted with 7 per cent of pitch E. The briquettes were well pressed, of a grayish color, but on cooling crumbled decidedly. They weighed 6.73 pounds each. As they did not contain an excess of pitch, 7 tons more of this coal were briquetted with 8 per cent of pitch E, in order to have a sufficient quantity for a steam test.[a] The resultant briquettes were bluish black in color, but they were not quite hard enough, although fairly strong, and would stand considerable hard treatment in transportation. In burning they held together until consumed. They weighed, on an average, 6.77 pounds each. The eggettes made from this same mixture were stronger than the briquettes, had a polished surface, but were very brown in color. In the cook stove they burned very satisfactorily, without crumbling hardly at all.

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[a] For result of this test see p. 81.

### KANSAS.

*Kansas No. 2.*—Two tons of this coal were briquetted with 11 per cent of pitch X. The pitch of the mixture, in passing through the machine from the pug mill to the press, set so quickly that the briquettes did not receive the necessary pressure. The surface was rough, and a great many cracks were developed. No more of this coal was available for briquetting after receiving pitch of the better qualities. It is, however, a coal that will briquette very readily with pitch of the better grades, such as pitch D or pitch H. It requires about 7 per cent of either for commercial briquettes.

### KENTUCKY.

*Kentucky No. 1.*—This coal is a good coking coal, but is unusually light in weight and is very lustrous. In passing through the disintegrator it was ground a little too fine, even when using only one side of the machine. Then, again, on account of its light weight it was difficult to give it the right pressure on the English machine. The coal was tested with 6 per cent of pitch D, and the resultant briquettes were clean and black, but not sufficiently pressed to close all the pores. They were, however, very good briquettes, and would stand rough treatment in transportation. They stood weathering with but little change. They weighed on an average 6.15 pounds each. The eggettes made from this same mixture were very smooth and lustrous and as strong as any eggettes that have been made on the American machine. In the cook stove they burned very satisfactorily without disintegrating or giving off an excess of smoke. This coal can be briquetted very readily.

*Kentucky No. 2.*—As there were none of the good pitches available at the time this coal was tested, it was briquetted with 9 per cent pitch E, which was the percentage determined by laboratory work. Three tons of this coal were tested with 9 per cent pitch E. The resultant briquettes were extremely pitchy, and on account of the excessive water due to wet steam, these briquettes were difficult to handle, and when piled three or four deep they crushed under their own weight. With dried steam the briquettes were more easily handled. Although still somewhat plastic, they did not crush when piled up. On cooling, the briquettes were very hard, having a lustrous fracture, and were capable of standing a great deal of rough handling. These briquettes weighed on an average 6.83 pounds each. The eggettes made of this same mixture were very hard and sonorous, but rather too brittle. In the cook stove the eggettes burned very satisfactorily.

This coal was tested with 8 per cent pitch E, using 3½ tons of coal and giving the briquettes the greatest pressure possible on the English machine.[a] As the coal is very hard, only one side of the disintegrator was used, and in the resultant briquettes there were many large pieces of coal, but there were also sufficient small pieces to fill in all the openings between the larger fragments. This is a nearly ideal condition for briquetting on the English machine. The briquettes were bluish black and excellent in every way. They did not have the glossy or lustrous fracture of the briquettes containing 9 per cent of pitch, but they were a better commercial briquette and were more easily handled when taken from the machine. With pitch of the quality of pitch D or pitch H, from 5 to 6 per cent would make a good briquette. These briquettes would stand a great deal of rough treatment in transportation, and in burning they did not disintegrate, nor was there much slack formed in breaking them for burning. They weighed on an average 6.83 pounds each and had a specific gravity of 1.13, as compared with 1.37, the specific gravity of the slack or fine coal. The slack which was used in making the briquettes contained 18 per cent of ash, but the lump coal contained only 11 per cent of ash. The crushing strength of the briquettes was 11,300 pounds to the square inch. The eggettes made from this same mixture were stronger and better than those made with 9 per cent, although not so sonorous and hard. They were also browner in color. The eggettes were not so strong as the briquettes.

## MISSOURI.

*Missouri No. 1.*—At the time this coal was tested only the hard pitch was available. Two and a half tons of this coal that had been previously washed were briquetted with 11.5 per cent of pitch A. The resultant briquettes were black in color and sufficiently hard to stand rough handling, but on account of the pitch setting too quickly they were insufficiently pressed and were granular and porous.[b] This coal will, however, briquette very readily, and with the softer pitches, as pitch D and pitch H, will make a good briquette with about 6 to 7 per cent of binder.

## MONTANA.

*Montana No. 1.*—This coal was dried slightly before briquetting. It was tested with 16 per cent of pitch A. The full limit of pressure that could be obtained on the English machine was used, and although the briquettes were well pressed, they were somewhat porous and rough on the surface. They were black, but not very glossy. They stood rough handling, and they weighed on the average 6.30 pounds each.

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[a] For results of steam test see p. 81.

[b] For steam test see p. 81.

This coal was briquetted with 7 per cent pitch B, using the highest pressure obtainable. As received from the press, these briquettes were somewhat plastic, but on cooling they proved to be brittle. They weighed on an average 7 pounds each.

## NEW MEXICO.

*New Mexico No. 1.*—This black lignite was first briquetted with 12 per cent pitch A, but the resultant briquettes were unsatisfactory in every way. Five and a half tons of this coal were briquetted with 15.8 per cent pitch X, and the resultant briquettes were apparently good in every way and could be handled very easily without any crumbling. On cooling, however, they began to crumble, and they did not stand the weather very well. In burning they showed a great tendency to disintegrate.

This coal was tested on the English machine with 8 per cent pitch D. The resultant briquettes were rather crumbly and earthy, and on cooling were covered with white efflorescence. They did not contain a sufficient quantity of pitch. They weighed on an average 6.83 pounds each and had a specific gravity of 1.13. Their crushing strength was 7,120 pounds to the square inch.

As 8 per cent of pitch D did not give any excess of binder, and as the resultant briquettes were not satisfactory, another ton of this coal was tested with 10 per cent of pitch D. While the briquettes fresh from the machine were somewhat stronger than those with 8 per cent of pitch D, it was found that upon standing and becoming thoroughly cool they became so like those made with 8 per cent pitch that they could not be distinguished from each other. This indicated that a satisfactory briquette could not be made with this pitch, even by greatly increasing the percentage. In making these briquettes, the English machine was run under ideal conditions, having a constant pressure; the coal was sufficiently dry and not too fine, and could be briquetted at a moderate temperature. There was a little more white efflorescence on the briquettes with the 10 per cent than with the 8 per cent pitch D, although the briquettes were quite glossy. The average weight of these briquettes was 6.42 pounds each and their specific gravity was 1.02. The crushing strength of these was 5,900 pounds per square inch, as compared with 7,120 pounds per square inch for those made with 8 per cent pitch D. The eggettes were quite glossy on the surface, but had a dull fracture and were not very tenacious. There was a tendency to disintegration in burning.

The most satisfactory test was with pitch H, and one ton of this coal was briquetted with 8 per cent of this pitch. The resultant briquettes were hard, strong, and clean, being much better in every way than any briquettes of this coal made with pitch D. They weighed on the average 6.25 pounds each and their specific gravity was 1.06. Their crushing strength was 13,050 pounds per square inch, nearly twice the strength of those made with 8 per cent pitch D.

*New Mexico No. 2.*—This black lignite is almost identical with New Mexico No. 1. It was tested with 9.3, 10.5, and 12 per cent of pitch X, but in every case the resultant briquettes were very crumbly and no positive results could be obtained.

This coal was next tested with pitch D as a binder, one ton of the coal being briquetted with 7 per cent of pitch D. The resultant briquettes were very crumbly, gray in color, and had an earthy fracture. They were neither hard nor strong and were porous, although the pressure was increased to the limit. There did not seem to be any difference in the resultant briquette, whether made with a low or high pressure. There was a large amount of grayish efflorescence that came out on the briquettes almost immediately. They weighed on an average 7 pounds each. The eggettes made from this mixture, although not so porous, had the same earthy appearance and fracture and gray color.

Another ton of this coal was briquetted with 9 per cent of pitch D, and there was an excess of pitch, as was indicated by the bluish color on the external surfaces, due to partial volatilization of the pitch. On account of an excess of steam, the briquettes were rather soft as received from the machine, and owing to the excess of pitch many of them stuck together. Although they were better than those made with 7 per cent pitch D, they were not commercial briquettes. They weighed on an average 7.12 pounds each. This coal was very dirty and contained considerable clay, and the same difficulties were experienced in briquetting it with pitch as with the Indian Territory No. 6 coal. From the work that was done on this sample, it has been definitely proved that such dirty lignites can not be well briquetted with any commercial percentage of pitch as a binder, unless perhaps it is a pitch made from petroleum, as pitch H.

Five tons of this coal were briquetted on the American machine, using 10.25 per cent of the Hoffman patent binder, consisting of petroleum, rosin, and quicklime. These briquettes were not very tough, but held together pretty satisfactorily when burned under the boiler.[a]

### NORTH DAKOTA.

*North Dakota No. 1.*—This is a tough, woody, brown lignite, which does not disintegrate very readily. In the first test the lignite was not previously dried and was mixed with 10 per cent of pitch A.

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[a] For steam test see p. 82.

Although the highest pressure possible was used, the resultant briquettes were very porous and had the appearance of an incoherent mass of chips. With 12 per cent of pitch A the briquettes were still porous and noncoherent, and on standing for some time exposed to the weather the grains and fragments of lignite began to slack off. Between the fragments of lignite there was an excess of pitch observed, but it did not seem to have any adhesion for the fragments. As the briquettes were cooling it was noticed that the contained steam acted upon the lignites, giving them the appearance of lumps of soft, brown loam surrounded by pitch.

The next test was made with dry material, the lignite having been previously ground to about 4-mesh. One ton of this dry, fine material was mixed with 12 per cent of pitch A and run through the English machine, only one side of the disintegrator being used. There was no excess of steam and the full pressure was used, but the resultant briquettes were open. With both sides of the disintegrator in use there was a slight improvement in the briquettes, but they were all a porous, incoherent mass, and on exposure to the weather began to disintegrate. There was apparently an excess of pitch between the flakes, similar to that observed in the first ton tested. The same mixtures were tested on the American machine, and the eggettes, when first received, had a polished, lustrous appearance, but became rough immediately, and they had almost no coherence. This lignite was tested further by increasing the percentages of pitch, allowing the mixtures to remain longer in the pug mill, and increasing the pressure to the limit that could be attained on the English machine, but no briquettes were obtained that had any coherence, and all the results were negative.

### PENNSYLVANIA.

*Pennsylvania No. 3.*—This sample consisted of a car load of anthracite culm. The first test was with 90 parts of the culm and 10 parts of a West Virginia coking coal, and 12½ per cent of hard pitch A. The mixture was run through the English machine, both sides of the disintegrator being used, and the briquettes being pressed to the limit of the machine. On account of the hardness of the anthracite coal, it did not disintegrate to any great extent and the resultant briquettes resembled concrete. These briquettes were hard and tough, and even with this inferior pitch were of good quality. They weighed on an average 7.25 pounds each. During combustion they burned like lumps of anthracite coal, making but little flame. On standing exposed to the weather the briquettes did not suffer to any extent except during hard rains, when their surfaces became pitted. Apparently this pitting was due to the wearing away of the softer pitch from the harder fragments of anthracite.

This culm was next tested with the softer pitch B, using 84 per cent of anthracite, 9 per cent of West Virginia coking coal, and 7 per cent of pitch B. For this test the coarsest and driest portion of the anthracite culm was used. This combination made fair briquettes, but they would have been improved if dry steam could have been used. The briquettes weighed on an average 7.64 pounds each and had a specific gravity of 1.174, while the specific gravity of the raw culm was 1.52. The crushing strength of the briquettes was only 4,500 pounds to the square inch. The eggettes of this mixture made on the American machine were harder and firmer than the corresponding briquettes. They all burned like anthracite coal, except that in starting the fire they were more easily ignited and burned at first with a short flame.

Another ton of the anthracite culm, which had been kiln-dried, was briquetted with 7 per cent of pitch B without the addition of any bituminous coal. If the steam introduced into the pug mill was wet, the fresh briquettes were almost too soft to handle. If, however, the steam was dry, they were firm and polished. Although the briquettes were not so lustrous as those made with soft coal, they were clean, smooth, hard, and strong, and when dropped they rang like stone. The eggettes of this mixture made on the American machine were also very good, but no better than the briquettes. Both the eggettes and the briquettes burned like lump anthracite, with very little flame or crumbling. In breaking the large briquettes there was but little waste. These weighed on an average 7.4 pounds each and had a specific gravity of 1.28, the specific gravity of the coal being 1.52. The crushing strength of the briquettes was 17,100 pounds per square inch, and the test was made on briquettes that had been exposed to the weather for over a month.

One ton of the anthracite culm was also tested with 7 per cent of pitch D as a binder; but, on account of lack of pressure, the resultant briquettes were porous and not so good as those which contained 7 per cent of pitch B. They stuck together while hot, indicating that 7 per cent of pitch D was sufficient and that an increase of pitch would not improve them. They weighed but 7.06 pounds each on an average.

A 5-ton lot of the anthracite culm was briquetted with 10 per cent of West Virginia coking coal and 11¼ per cent of the Hoffman patent binder on the American machine. The resultant eggettes were hard, smooth, and clean, with a somewhat polished surface, and stood considerable rough handling. These eggettes were tested by burning under a boiler, the fire box of which was fitted for a bituminous coal.[a] Although made of 79¾ per cent of anthracite, they did not burn like anthracite, and their oil and rosin constituents gave a long flame to the fuel and the boiler ran at 86 per cent of its rated capacity.

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[a] For result of steam test see p. 82.

From the results of these tests it is apparent that anthracite culm can be satisfactorily briquetted either in combination with a coking coal or by using simply pitch as a binder, and that the resultant briquettes have most of the properties of lump anthracite.

## WEST VIRGINIA.

All of the West Virginia coals tested at the plant were good coking coals, so that there was little need of briquetting experiments.

*West Virginia No. 3.*—A test was made with this coal in the laboratory to determine if it is possible, under a certain pressure and temperature, to make a briquette without any binder. A briquette was obtained which, although not perfectly satisfactory, indicates that under certain conditions, with the right pressure and temperature, briquettes could be made with this coal without any binder whatever.

*West Virginia No. 6.*—The first test made on this coal was with 5 per cent of the very glossy homemade pitch Y. The pressure used was the limit that could be obtained on the English machine, and the resultant briquettes were hard, clean, and strong, and had a sharp glossy fracture. They were in every way equal to the best briquettes that had been made on this machine. They weighed on an average 6.12 pounds each. On account of the excellent quality of the briquettes with 5 per cent of pitch, another ton was briquetted with only 3 per cent of pitch Y. Extreme pressure was applied, and the resultant briquettes were unusually smooth and glossy, but not quite strong enough. They crumbled instead of cracking clean under the blows of a hammer. The briquettes containing 5 per cent of pitch, when broken, made little or no waste and had a clean fracture.

On the American machine the mixture containing 5 per cent of pitch Y made very glossy and solid eggettes, which were very strong and would stand a great deal of very rough handling. One surprising quality of these briquettes was that when tested in the cook stove they burned with little or no caking, although the coal itself is a very good coking coal. The briquettes, when exposed to the weather, were affected but little and did not disintegrate appreciably. In burning they held together until entirely consumed.

This coal was also tested with the Hoffman patent binder, and the resultant eggettes were also of very good quality, being hard, glossy, and clean; but in burning they caked more than those with 5 per cent of pitch.

## WYOMING.

*Wyoming No. 1.*—This coal was a very black, pitchy-looking lignite, which contained 23 per cent of water. It was first tried without drying, 9 per cent of pitch H being used. Judging from the resulting briquette, there was a slight excess of pitch, but on account of the large amount of water in the lignite the pitch apparently failed to adhere to the grains of lignite. The fresh briquettes were difficult to handle, and when broken open the grains of lignite seemed to be wet, as if coated with moisture, which prevented the pitch from adhering to them. Many of the briquettes were porous, not being sufficiently pressed, and they were also brittle. Some of them, however, were quite good. When broken up and burned in the stove they stood up in the fire very well and burned better than any lignite briquette that had thus far been made. They weighed on an average 6.31 pounds each.

Another ton of this lignite was ground in the Williams mill and run through the drier, but this operation removed only a portion of the water, so that the briquette still contained considerable water. This material was briquetted on the English machine with 8 per cent of pitch H. A minimum amount of steam was used, which did not contain any excess of water, and a maximum pressure was applied. As received from the machine, many of the briquettes showed perpendicular cracks due to the excessive pressure. The hot briquettes were difficult to handle, the individual grains of the lignite seeming to be wet and not cohering at all. Even under their own weight many of them would crack badly. On breaking open the hot briquettes the grains of lignite seemed to move about as if they were alive. This lack of cohesion continued until the pitch became quite hard and could no longer be pressed in the machine. Thus the variation in the steam used as the mixture passed through the pug mill had no effect on the cohesion of the resultant briquettes. On cooling, however, these briquettes became very hard and strong and broke with a sharp fracture and with but a small amount of waste. In burning, these briquettes did not fall to pieces, as had been the case with the other lignite briquettes, but burned very satisfactorily. They were very light, weighing only 5.85 pounds each, and had a specific gravity of 0.98. The specific gravity of the coal was only 1.16. The crushing strength of these briquettes was 9,600 pounds to the square inch.

Another ton of this coal was tested with a soft, tough asphalt, B6, that was received from Casper, Wyo. As had been shown by laboratory tests, this asphalt could not be used on the English machine alone, and as a result the lignite was briquetted with 5 per cent of asphalt B6, 2½ per cent of rosin, and 1 per cent of lime which had previously been hydrated. This mixture worked perfectly in the machine, and the hot briquettes were firm and good and easily handled, but the proportion of binder was not sufficient and the briquettes were dry and not strong. Another reason for this result was that during the two crushings and dryings of the lignite the grains had become of a nearly uniform size (about 10-mesh), so that the resultant briquettes were very porous in spite of the maximum pressure used. In burning, the briquettes fell to pieces. They weighed on an average 5.40 pounds each. A combination of this coal and asphalt can undoubtedly be obtained which will briquette satisfactorily.

## COKE BREEZE.

In order to determine whether it was possible to use waste coke breeze, a series of experiments was made in the laboratory which showed that coke breeze could be briquetted so that it could be used as a substitute for lump coke. In order to make a practical test, a ton of this mixture was made containing 73 per cent of crushed coke, 20 per cent of West Virginia No. 6 coal, and 7 per cent of homemade pitch Y. These materials were mixed thoroughly by hand and run through the English machine, both sides of the disintegrator being used. They were well pressed, and as delivered by the machine they were unusually dry and had a brassy to bronze sheen on the surface, which looked as though it might be due to an abrasion of the brass of the molds. Upon testing this material, however, it was found to contain only the faintest possible trace of copper. The briquettes were hard and strong and easily handled while hot, but were somewhat dirty. On cooling they became unusually tough and strong, and during combustion were very solid and burned more like anthracite than bituminous coal. Such a combination would make a good substitute for anthracite coal. These briquettes weighed on an average 5.73 pounds each.

The same mixture was run through the American machine, and the resultant eggettes were dense, smooth; and polished, but not so tough as the corresponding briquettes. While warm they would stand a great deal of rough handling, but on cooling they became more brittle. This may be due to the crushing of the coke grains by the excessive pressure obtained in the American machine.

The next test of coke breeze was made without the introduction of any bituminous coal, and the mixture consisted of 92 per cent of coke breeze and 8 per cent of pitch D. In passing this mixture through the English machine both sides of the disintegrator were run, a minimum amount of steam was used, and a moderate pressure exerted. The resultant briquettes were clean, well formed, and when struck had a metallic ring. They were very tough and strong, and would bear much more handling than any of the briquettes made out of the coal. They burned like ordinary coke, without any disintegration and with only a little flame. These briquettes weighed on an average 5.92 pounds each. The well-pressed briquettes had a specific gravity of 1.025 and the porous briquettes had a specific gravity of 1.002. The crushing strength of the better briquettes was 30,100 pounds to the square inch, while the porous briquettes had a crushing strength of 22,100 pounds. In comparing the results of the crushing strength of these coke briquettes with those made from coal, it will be seen that even the porous briquettes had a much greater crushing strength than the best of the coal briquettes.

The eggettes of this same mixture that were made on the American machine were very hard and had the same metallic ring as the briquettes. On account of the fragments of coke being crushed in the press, these were denser than the briquettes, but not so tough. They were, however, very satisfactory. The manufacture of these eggettes or briquettes should be a means of utilizing a considerable amount of waste coke.

## COKING OF BRIQUETTES AND BRIQUETTE MIXTURES.

In connection with the briquetting of the various coals, a line of experimental work suggested itself, viz, the possibility of making a coke out of a semi to non coking coal by the introduction of sufficient volatile matter in the form of a binder to cause the coal to coke. The first experiment along this line was tried with some of the briquettes made from Arkansas No. 4 coal and 6 per cent of pitch B, by filling a box with about 12 crushed briquettes and then placing this in one of the coke ovens, where it remained during a run of a coking coal. The result of this experiment was the obtaining of a mass of coke which was 12 to 15 inches long, somewhat metallic in appearance, and had a good ring to it.[a] The next experiment was the filling of a box with a mixture of Arkansas No. 4 coal with 6 per cent of pitch B, but without passing it through the press. This was treated in the coke oven similarly to the other, but the results were not so satisfactory as in the first case. A coke was obtained, but it was not so good as the other. The next experiment was with the same mixture after briquetting but without breaking up the briquettes. These were piled on top of one another in a box and tested in the coke oven in the same manner as the others. The result of this test was the obtaining of masses of coke that retained the form of the original briquette. All of the coke obtained by these experiments, while not of the quality desired for iron smelting, could be used for lead and copper smelting.

Another experiment on a larger scale was with 2 tons of Arkansas No. 6 slack coal. This coal was mixed with 8 per cent of pitch A and then ground in the Williams mill and charged into a coke oven together with an equal weight of the same mixture that had been previously briquetted. Both mixtures made coke, but not of the best quality. On account of the briquettes not being packed close together, a portion of them were somewhat burned and were not so good as the remainder.

The experiments seem to show that while a coke can be made by mixing coal and pitch, coke of a better quality is obtained by previously briquetting the mixture; but sufficient experiments along this line have not yet been made to justify any definite conclusions, and it is expected that the experimentation will be continued.

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[a] For coking test see pp. 130-131.

|  |  |  |  |
|---|---|---|---|
| y | do | do | [illegible] |
|  | do | do | 5,900 |
|  | Good | Good | 13,050 |
|  | Disintegrates | Disintegrates |  |
|  | Fair | Fair |  |
|  | Disintegrates | Disintegrates |  |
|  | do | do |  |
|  | Becomes pitted | Good, like anthracite |  |
|  | do | do | 4,500 |
|  | Good | do | 17,100 |
|  | Very good | Very good |  |
|  | Good | Good |  |
|  | Fair | Satisfactory |  |
| and | Good | Good | 9,600 |
|  | Fair | Fell to pieces |  |

te.

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DEPARTMENT OF THE INTERIOR
UNITED STATES GEOLOGICAL SURVEY
CHARLES D. WALCOTT, DIRECTOR

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# CONTRIBUTIONS TO MINERALOGY

FROM THE

# UNITED STATES GEOLOGICAL SURVEY

BY

F. W. CLARKE, W. F. HILLEBRAND, F. L. RANSOME, S. L. PENFIELD, WALDEMAR LINDGREN,
GEORGE STEIGER, AND W. T. SCHALLER

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# LETTER OF TRANSMITTAL.

DEPARTMENT OF THE INTERIOR,
UNITED STATES GEOLOGICAL SURVEY,
*Washington, D. C., January 31, 1905.*

SIR: I have the honor to transmit herewith a series of papers by various hands for publication as a bulletin. They represent mineralogical investigations incidental to geological researches and important to the progress of geology, and are here brought together for convenience of reference.

Very respectfully,
G. F. BECKER,
*Geologist in Charge Division of Chemical and Physical Research.*

Hon. CHARLES D. WALCOTT,
*Director United States Geological Survey.*

# CONTRIBUTIONS TO MINERALOGY FROM THE UNITED STATES GEOLOGICAL SURVEY.

## PREFATORY NOTE.

By F. W. Clarke.

Although mineralogical research occupies a relatively small place among the many varied activities of the United States Geological Survey, it is by no means neglected. From time to time interesting material is collected by the field parties of the organization, and specimens deserving study are received from other sources. These are worked up, as opportunity offers, in the chemical laboratory of the Survey, sometimes independently and sometimes, as in the work reported in two of the memoirs here presented, in cooperation with the geologists by whom they were brought in. Occasionally assistance is received from investigators not connected with the Survey, and one such case is now represented by the joint paper of Doctor Hillebrand and Professor Penfield, of Yale University. In that instance the chemical work was done in Washington and the crystallographic measurements were made at New Haven.

Most of the papers contained in this bulletin represent investigations which are complete within themselves. The work done by Mr. Steiger upon the silver and thallium derivatives of the zeolites, however, is to be regarded as one part of a series of researches upon the constitution of the silicates. Some parts of this series have been already published, and it is hoped that this work may be continued in the future. The field to be covered is enormous and progress within it is necessarily slow. Still, the results obtained so far are definite and significant, especially as they show a chemical plasticity among the silicates which was formerly unsuspected.

Some of the items of more than ordinary interest to be found within these pages are as follows: Three new mineral species—morencite, coronadite, and plumbojarosite—are described. Natrojarosite, although not absolutely new, is now thoroughly established as a definite mineral. The true character of carnotite is shown in Doctor Hillebrand's research, and the same chemist has worked out more fully than his predecessors the composition of yttrialite and its relations to other species. Mr. Schaller's very complete study of dumortierite is also worthy of especial mention, and his notes upon other minerals from California are but the first fruits of a much larger research which is already well under way.

# ON CARNOTITE AND ASSOCIATED VANADIFEROUS MINERALS IN WESTERN COLORADO.

By W. F. Hillebrand and F. L. Ransome.

## INTRODUCTION.

By W. F. Hillebrand.

The rather widespread occurrence in western Colorado of considerable quantities of a canary-yellow ore of uranium has been known for a few years past. It has been marketed to the extent of a few tons, but its mineralogical nature was unknown or incorrectly surmised until 1899.

In the spring of that year a specimen of this ore first reached my hands through Dr. George P. Merrill, of the United States National Museum, and was soon found to contain a mineral or minerals new to science. Within a week appeared the announcement by MM. C. Friedel and E. Cumenge[a] of a new mineral, carnotite, a hydrous vanadate of uranium and potassium, obtained through Mr. Poulot of Denver, from Roc Creek, Montrose County, Colo. Mr. Poulot had already identified vanadium in it. It was at once seen, despite certain differences in composition, that the two were identical.

According to the French authors the mineral is of simple composition, as above expressed, with only a little iron and mere traces of Al, Ba, Cu, Pb, and also, according to M. and Mme. Curie, of the radio-active substances radium and polonium. The empirical formula $2U_2O_3$, $V_2O_5$, $K_2O$, $3H_2O$, was assigned to it, some doubt attaching to the water.

Since then I have been able to examine carnotite from several localities in western Colorado, finding in each case the same lack of agreement with the analyses of Friedel and Cumenge.

While I was engaged in this work my attention was called to certain more or less greenish sandstones from the vicinity of Placerville on the San Miguel River, San Miguel County, Colo., which were said to be highly vanadiferous and of considerable extent, and in which a zone a few inches thick was rather strongly impregnated with a yellow mineral resembling and probably identical with carnotite. This mineral appeared also scattered through the sandstone at other points in sporadic small patches, sometimes visible only by aid of a lens.

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[a] Bull. Soc. Chim. de Paris (3), xxi, 328, 1899; Bull. Soc. Franc. Min., xxii, 26, 1899; Comptes Rendus, cxxviii, 532, 1899; Chemical News, lxxx, 16, 1899. The papers as published in French differ slightly.

This occurrence led to the thought that the carnotite bodies farther west might also be associated with existing or dependent on preexistent vanadiferous sandstones. For the carnotite of Montrose and Mesa counties, as is stated by the French authors, occurs mixed in all proportions with quartz-sand grains, the remnants, beyond doubt, of former sandstone bodies, and it was soon recognized that in the carnotite bodies the vanadium existed in two conditions, the larger part by far as pentavalent vanadium in the easily soluble carnotite, and a smaller and sometimes hardly distinguishable portion as trivalent vanadium in a much less soluble silicate which was free from uranium.

It was hoped that, by a field reconnaissance, observations bearing on these points would be obtained and material assembled which would on analysis help to determine the nature and explain the association of these two entirely different classes of mineral substances.

The field and microscopical observations of Messrs. Ransome and Spencer are embodied in the pages immediately following these introductory remarks. Unfortunately their collections, except those from Placerville, came to hand so late that the chemical work on the more western occurrences has been confined to the carnotite bodies alone, specimens of which were already in my hands. Greenish sandstones have been observed by Messrs. Ransome and Spencer in those regions, but whether any of them are highly vanadiferous, or what their connection with the carnotite may be, remains yet unknown. The only two examined did not owe their color to vanadium. Meanwhile the chemical results thus far obtained will be found in the concluding section of this paper.

In this place it is my pleasure to record my appreciation of the readiness with which the following gentlemen have supplied me with material for study: Messrs. Poulot and Voillequé of Denver, Mr. A. B. Frenzel of Placerville, and Mr. J. R. Duling of Paradox. These gentlemen have also not hesitated to give me all information at their disposal as to occurrence and field relations of these interesting ore bodies.

## OCCURRENCE OF THE URANIUM AND VANADIUM ORES.

By F. L. Ransome.

### GENERAL STATEMENT.

In the autumn of 1899, accompanied by Dr. A. C. Spencer, whose knowledge of the stratigraphy of the region was of great assistance, I made a hasty reconnaissance trip into the western portions of *San Miguel, Montrose,* and Mesa counties, near the Utah-Colorado line. The primary object of the expedition, in accordance with a suggestion from Mr. S. F. Emmons, was to investigate the copper deposits of La Sal Creek, Paradox and Sinbad valleys, and the vicinity, which had been the occasion of some mining excitement a few months before. A memorandum from Dr. W. F. Hillebrand, received just before starting, indicated that it would be well also to examine certain prospects on which some preliminary work had been done, looking toward the extraction of ores of uranium and vanadium. It is to these ores that the present notes are confined.

The most convenient way of reaching Paradox Valley was found to be the stage road, which, starting from Placerville, a settlement and station on the Rio Grande Southern Railroad, runs by way of Norwood, Shenandoah, and Naturita to Paradox, a distance of 60 or 70 miles. The region can also be reached from the west by way of Moab, in Utah.

The topography of the region west of Placerville is that characteristic of the "mesa country" of western Colorado and southeastern Utah. Broad stretches of plateau are intersected by steep-walled canyons and cliff-encircled valleys. The underlying rocks comprise the "Red Beds" of the Dolores formation[a] (Triassic), the La Plata formation (Jurassic), the McElmo formation (Jurassic), the Dakota sandstone (Cretaceous), and the Mancos shales (Cretaceous). Carboniferous rocks occur in Sinbad Valley, and a series of gypsum-bearing shales of unknown age is found in the bottoms of Sinbad and Paradox valleys, but as the ore deposits to be described all occur in the beds of the La Plata and McElmo formations, these older rocks need not again be referred to. The sediments making up these various formations lie usually nearly horizontal, but they are sometimes flexed and frequently faulted.

In all of the prospects examined the ore of uranium occurs in the form of the recently described bright-yellow carnotite. In one case this is intimately associated with a dull olive-green mineral which, according to Doctor Hillebrand, is either identical with, or very closely allied to, the vanadium-mica roscoelite. Deposits of one or both of these minerals occur widely scattered over San Miguel and Montrose counties, Colo., and in the Blue Mountain (Sierra Abajo) district of southeastern Utah; but only a few of the known deposits were personally examined.

## PLACERVILLE DEPOSITS.

These are essentially vanadium deposits, and occur 4,500 feet nearly northeast of the railway station near Placerville, and about 1,000 feet above the San Miguel River. The lower 900 feet of the San Miguel canyon is cut in the characteristic "Red Beds" of the Dolores formation. Above the Dolores comes the La Plata sandstone in typical development, as described in the Telluride folio, viz, two heavy beds of light-colored sandstone, separated by a much thinner bed of dark limestone. The roscoelite occurs as an impregnation in the lower bed of the La Plata sandstone, about 100 feet above the base, and just beneath the bed of dark limestone. All the beds at this point are practically horizontal. The dark green vanadiferous sandstone occurs in a nearly continuous band, approximately parallel to the bedding planes, and varying in thickness from a few inches up to 5 or 6 feet. This band extends along the sandstone cliffs for an estimated distance of about 2,000 feet. The roscoelite more or less thoroughly impregnates portions of the fine-grained sandstone which constitutes the mass of the bed. It sometimes makes up more than 20 per cent of the vanadiferous facies. Normally the sandstone of this lower bed of the La Plata is light buff in color, with patches, mottlings, and stripes of pale pink, the latter color being apparently due to small amounts of ferric oxide. But when richly impregnated with roscoelite this light-colored sandstone becomes dark olive-green, the contrast being most marked when the cliffs are wet. Although fairly continuous for the distance stated, this zone of vanadiferous sandstone is by no means regular. It varies much in thickness, and in one place splits into two or more branches. At some points two or more distinct streaks of roscoelite-bearing sand were found at different horizons in the main sandstone bed. The carnotite is not nearly so abundant as the roscoelite at this locality. It occurs as minute yellow specks in the sandstone, and particularly as thin horizontal seams or streaks near the bottom of the vanadiferous zone.

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[a] The formation names used in these notes are those adopted by Mr. Whitman Cross in the text of the Telluride folio (folio 57), of the U. S. Geological Survey. The reader is referred to this folio for fuller descriptions.

The work thus far done on the several claims which have been located along these deposits is of the most superficial character. On the Canary claim Mr. A. B. Frenzel has run a tunnel, about 18 feet long, which exposes a typical section of the impregnated zone. The roof of the tunnel is formed by the under side of the bed of dark limestone previously referred to. This limestone is underlain by a few inches of sandy limestone which passes into the light-buff La Plata sandstone without break. The latter contains abundant calcite as a cement or matrix for the sand grains. At a distance of a foot below the limestone, the sandstone shows, on close inspection, numerous specks of carnotite and gives a qualitative reaction for vanadium (Hillebrand). Examined in thin section under the microscope, this portion of the bed shows a fine-grained homogeneous sandstone, in which well-rounded grains of quartz are held together by a rather abundant matrix of calcite. The calcite is crystallized as a fine granular aggregate and includes small indeterminable particles of various kinds. Many of the quartz grains, as seen in section, are entirely surrounded by calcite, and so isolated from adjacent grains. A pale lemon-yellow substance occurs sporadically in the section as an interstitial material between the quartz grains. This is probably the carnotite, which was seen as small, yellow specks in the hand specimen. It is very minutely crystalline, recalling the habit of some obscure chloritic aggregates in decomposed igneous rocks, and is too indistinct for successful optical study. Small crystal grains of zircon are scattered through the sandstone, and are readily concentrated in the residue after treating with hydrofluoric acid.

Within the next 2 feet below the zone where it exhibits the foregoing facies, the sandstone becomes pinkish in consequence of the presence of ferric oxide, and in this pinkish rock the lens fails to reveal any roscoelite or carnotite. But just below the pink zone the sandstone begins to show yellow and green specks. The latter especially become more numerous and larger, until at from 3 to 4 feet below the limestone the sandstone has a decided green tint. Farther downward this deepens, until near the floor of the tunnel the sandstone is a uniform deep olive-green. This green rock is rich in roscoelite (over 20 per cent) and shows many small yellow specks of carnotite. This is regarded as the first-class ore, and the chief value of the deposit is supposed to be in its vanadium, the uranium being very subordinate. In this respect the Placerville deposit differs from those on La Sal and Roc creeks, presently to be described, where the mineral sought for and occurring most abundantly is carnotite.

Thin sections of the green-spotted sandstone, when examined microscopically, resemble those of the light-buff sandstone just above it, as far as character of quartz grains and relative abundance of matrix are concerned. But the character of the matrix or cement is different. Calcite is much less abundant and its place is largely taken by roscoelite. This is grass-green in thin section, and might readily be mistaken for indistinct wisps and areas of chlorite. It sometimes forms a distinct envelope around the quartz grains, showing an indistinct foliated or fibrous structure, with the fibers normal to the surface of the grains.

The uniformly dark-green sandstone, richest in roscoelite, does not effervesce with acids and shows no calcite cement in thin section. The usual quartz grains are held together by roscoelite. This is crystalline, but the highest available powers show only an indistinct and minute foliation, such as may be observed in some very finely crystallized chlorites.

Close to the floor of the tunnel is a fairly regular, nearly horizontal streak of carnotite, varying in width, but usually less than an inch wide, and showing noticeable diminution in thickness in the face of the tunnel. This small seam is not solid carnotite, but is merely a zone in the sandstone impregnated with the bright-yellow uranium mineral. It is not nearly so continuous as the main vanadiferous belt, and was seen only at three or four places along the outcrop of the belt, which extends for a distance of about 2,000 feet. When this seam is closely examined it is seen that narrow bands rich in carnotite alternate with green bands carrying mostly roscoelite. There is also usually present a seam of almost pure quartz, generally about an eighth of an inch in thickness, though often thicker. The microscope shows it to be a true quartzite, in which the original rounded detrital grains of quartz have been cemented by fresh quartz in optical continuity with the older granules. A similar quartzite occurs in the green vanadiferous sandstone above the carnotite, where it forms concretionary knots and nodules. It was not noted in the buff sandstone, however, where the cementing material is calcite.

Immediately below the carnotite seam there is a parting or "floor" in the sandstone, probably originally a very thin layer of shale, which forms the working floor of the tunnel. The sandstone below this floor is plainly impregnated for a short distance with roscoelite, but the thin shale seam is regarded as the practical bottom of the deposit.

The questions of the origin and actual extent of this deposit are closely related and are of much interest. Their discussion will be deferred until the other deposits visited have been described.

Some distance below the Placerville vanadium deposit a zone of green sandstone, presumably belonging to the Dolores formation, was observed. This color is a somewhat brighter green than that of the vanadiferous sandstone higher up the slope. As there are some copper prospects near by, from which ore has been taken, this was supposed to be a copper stain. Qualitative tests by Doctor Hillebrand show, however, that the color is due to a compound of chromium.

Similar green sandstones occur on the western side of Sinbad Valley in what is apparently the La Plata formation, and were originally supposed to be impregnated with roscoelite. Doctor Hillebrand's investigations, however, show that they too owe their color to some chromium mineral. These occurrences are interesting as showing that a green color in sandstones may result from various causes, and that even a bright green tint can not be taken as an infallible indication of copper.

### LA SAL CREEK DEPOSITS.

These occur in the extreme western portion of Montrose County, southwest of Paradox, and about 6 miles up La Sal Creek from Cashin. They are reached by trails from Paradox Valley and from Cashin. The deposits are on the south side of La Sal Creek and about 700 feet above the stream. They occur for a distance estimated at more than a quarter of a mile, along the sandstone cliffs which descend from the mesa into the canyon of La Sal Creek, and only a few feet below the level of the mesa surface.

In the absence of continuous stratigraphic work, it is impossible to correlate certainly and finally the rocks on La Sal Creek with the divisions established by Cross and Spencer in the Telluride quadrangle, which lies just east of this area. It seems probable, however, that the La Plata sandstone attains a much greater thickness in this portion of western Colorado than it does between Telluride and Placerville. The limestone bed, so characteristic a feature of the formation near Placerville and farther east, is not uniformly present in this western region, and the La Plata sandstone (Variegated beds, in part, of the Hayden survey) is not always readily differentiated from the underlying Dolores formation (Red Beds). For a vertical distance of about 400 feet above the bed of La Sal Creek the rock is a heavy-bedded, rather fine-grained, light-colored sandstone, which is considered by Doctor Spencer to be the La Plata. Above this come thinner-bedded sandstones, with some conglomerates and shales, which are included in the McElmo formation. It is in this upper series that the uraniferous deposits occur. All the beds are here approximately horizontal.

As revealed by numerous small openings near the crest of the bluff, the carnotite, which is the material here sought, is found chiefly in a massive bed of nearly white sandstone. Some of the ore, however, lies between the sandstone and a lower bed of light-gray shale. Although the prospecting openings all lie at about the same level along the cliffs, the deposit is not nearly so regular as the vanadiferous band near Placerville.

The carnotite of La Sal Creek occurs as irregular, bunchy "pockets" in the sandstone, or along the contact of the sandstone with the underlying shale. These have all the appearance of being impregnation deposits, the solutions carrying the uranium compounds having deposited the ore wherever they found ready passage through the rock—usually along bedding planes. No roscoelite was seen with the carnotite.

The most remarkable and interesting fact in regard to the La Sal Creek deposits is their very superficial character. The ore bodies are usually flat-lying streaks, a few inches thick, which grade above and below into the common light-buff sandstone, and which die out and disappear when followed into the hillside. In tunnels that run but a few feet underground the yellow impregnation of carnotite can be seen gradually to die out and to be succeeded by light-colored sandstone, showing no trace of the mineral. It is doubtful whether any appreciable quantity of carnotite occurs as much as 20 feet from the surface, on any of the locations, although this distance is given from memory and not from measurements on the ground. As before stated, the impregnation has usually taken place along bedding planes; it has also proceeded along surfaces of minor and superficial movement in the rocks. In one case it was observed that a portion of the overlying sandstone had moved upon the underlying shales, the disturbance being apparently a superficial one, of a kind commonly enough observed where massive beds rest on yielding shales on a steep hillside. In other words, the movement appeared to be directly related to the *present topography*. The carnotite was here deposited subsequent to the movement and had filled the small openings and dislocations in the shale caused by this very recent disturbance. It was reported that some of the best nests of ore had been found in "slide rock," i. e., rock which had slipped to some extent down the slope; but I was unable to verify this statement further than is indicated in the preceding description. There can be little doubt that the deposits of carnotite on La Sal Creek are not only very superficial in character but very recent in age.

### ROC CREEK DEPOSIT.

The principal claim is the Copper Prince, owned by J. R. Duling, on the north side of Roc Creek, 3 or 4 miles above its mouth, and near the foot of the Miller trail to Paradox. It is reached by this trail from Paradox Valley, or by trail from Hydraulic on the Dolores River. This deposit is in the La Plata sandstone, according to Doctor Spencer's observations, and has been more extensively exploited than any other seen. The sandstone, which at this point is nearly horizontal, is cut by an east-west fault, the fault plane dipping north at about 70°. The amount and character of the throw could not be determined. It is probably less than 50 feet. The carnotite occurs in the hanging wall of the fissure as small irregular branches in a loose mass of crushed sandstone and also as an impregnation of some of the firmer portions of the bed. No roscoelite was seen. Several small tunnels have been run in on the hanging wall side of the fissure (which itself carries no vein or ore-body), but they had been securely closed by planking, and no examination of their inner ends could be made. I am thus unable to tell what changes the deposit would show if it could be followed inward from the surface. A few hundred pounds of picked carnotite ore has been shipped from this claim and is reported to have sold for $1.25 a pound in Denver. This deposit is similar in character to others examined, except that in this case a well-defined fault has provided a zone of crushed and porous rock in the hanging wall, along which impregnation could take place. A few hundred feet farther west the crushed sandstone adjoining the fault has been impregnated with cupriferous solutions and is spotted with stains of the blue and green carbonates of copper.

### OTHER DEPOSITS.

It is known that several carnotite claims have been located in Gypsum Valley in what is known as the Disappointment district. The impregnated sandstone is said to cap a hill and to constitute an extensive deposit, but it was not visited. Carnotite is also reported from the Blue Mountain district, but I have no personal knowledge of these occurrences. It seems highly probable that the material will be found widely distributed in the Mesozoic sandstones of western Colorado and eastern Utah, although perhaps nowhere in very extensive bodies.

## ORIGIN OF THE DEPOSITS.

That the bodies of carnotite and roscoelite were formed subsequently to the deposition of the sandstones is evident from the facts presented in the preceding pages. It is equally plain that the minerals could not have resulted from the alteration, in place, of other compounds of vanadium and uranium originally contained in the sands. The shape and position of the deposits indicate clearly that the ores have been brought to their present position by transportation. Moreover, the recency of the deposits and the fact that they are sometimes directly connected with faults and dislocations in the sandstones show that the vanadium and uranium compounds could not have been the original cementing material of the quartz grains, but in all probability they have locally replaced the calcite that acts as matrix to the ordinary light-colored sandstone in which the ore bodies occur. The deposits of roscoelite appear to be comparable to impregnations formed in sandstones by cupriferous solutions—such impregnations as may be seen at many places in this region, particularly on La Sal Creek near Cashin and in Sinbad Valley—whereby the sandstone becomes colored bright green with carbonate of copper. In these places, however, the copper appears to have been previously deposited in part as chalcocite. An analogy might also be drawn with the green chromiferous sandstone near Placerville, already referred to. In all these cases the actual sources of the materials which have been deposited by solutions in their present position are not known. Doctor Hillebrand has shown,[a] however, that vanadium in small amounts is widely distributed in sandstones, limestones, and igneous rocks. It is perhaps present in very small amounts throughout the sandstone, and the deposits described may simply represent a concentration of this material under certain favorable conditions of solution and redeposition.

In the absence of exploitation it is manifestly impossible to predict the probable shape and size of ore bodies formed in this manner. The roscoelite seen near Placerville appears, however, to be much more persistent than the carnotite. There is no apparent reason why a mass of sandstone, impregnated with roscoelite, which is continuously exposed for several hundred feet along a cliff, should not extend for a considerable distance inward from the cliff face. The carnotite, on the other hand, appears to be of much more superficial occurrence, and its connection with the present surface of the ground is not yet fully understood. This would indicate that the carnotite results from a local concentration of material already existent in the sandstone, and the deposition of this material in the form of carnotite under conditions determined by proximity to the surface, and probably partly dependent upon a semiarid climate.

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[a] Am. Jour. Sci., 4th ser., vol. 6, 1898, pp. 209-216.

## CHEMICAL ANALYSES AND DISCUSSION.

By W. F. HILLEBRAND.

### I. GREEN COLORING CONSTITUENT OF PLACERVILLE SANDSTONE.

As noted by Doctor Ransome, the greener the sandstone the greater has been the replacement of the calcareous cementing material by the crystalline vanadiferous mineral. Fortunately, the greenest available specimen was entirely free from carbonate, and this was used for analysis, 10 grams being taken for the main portion.

The green substance was but slightly attacked by cold acids and not rapidly by hot ones, nevertheless after from twelve to twenty-four hours' digestion on the water bath with diluted nitric acid the greenish color had been transferred to the liquid, the residue of sand was nearly white, and a rather voluminous separation of nongelatinous silica had taken place. The filtration and washing of the insoluble matter was accomplished without difficulty. The free silica in it was extracted by two or three digestions of fifteen minutes each on the water bath with sodium carbonate solution of 5 per cent strength. The flocculent matter had entirely disappeared and the sandy residue was collected in a Gooch crucible and washed with hot sodium carbonate solution, followed by very dilute nitric acid, and finally by alcohol, to prevent turbid filtration. Drying of the sand was effected by allowing the pump to draw air through the crucible until no further loss in weight resulted.

The water in the sand was then determined at 105°, at 300°, and above 300°. Similar water determinations having been made on the unattacked substance, the differences gave the water expelled at those temperatures from the soluble part.

The sodium carbonate solution was acidified, evaporated, and filtered, and the filtrate again evaporated, to recover the last of the silica. To this was added a very small portion which the nitric-acid extract of the sandstone held in solution. The sum represents the total silica of the decomposed portion of the sandstone.

The further analysis of the nitric solution presented difficulty because of the vanadium it contained. It was carried out in a variety of ways, of which the following seemed to be most satisfactory:

Barium was first precipitated by sulphuric acid and subsequently separated from traces of lead, calcium, and vanadium. Hydrogen sulphide then threw out further traces of lead and copper with much sulphur from reduction of $V_2O_5$ to $V_2O_4$. After evaporation of the filtrate to reoxidize vanadium and iron, the alumina, iron, and trace of uranium with much vanadium were separted from the calcium, magnesium, and alkalies by three precipitations by ammonia solution. The combined filtrates were evaporated and ignited, the residue was transferred to a porcelain boat with nitric acid, evaporated therein to complete dryness, and exposed in a glass tube to a current of dry hydrochloric-acid gas, as recommended by Smith and Hibbs[a] for the expulsion of vanadium from alkaline vanadates. The removal of vanadium being incomplete, the contents of the boat were again evaporated with nitric acid and again distilled, and these operations were repated till no further evidence of a brown distilate appeared. The boat now contained only magnesium and alkalies with a little calcium. The distillates, containing all the vanadium that was in the boat, had been collected in U-tubes charged with water, and were eventually obtained in sulphuric solution for further treatment by evaporating with sulphuric acid in porcelain.

The precipitate of $Al_2O_3$, etc., was dissolved in nitric acid, evaporated almost to dryness in platinum, and boiled with sodium-hydroxide solution. These operations were repeated on the precipitate after filtration. The iron and titanium thus purified were redissolved, precipitated by ammonia, weighed, redissolved by acid potassium sulphate, and the ferric iron was reduced by hydrogen sulphide and titrated with permanganate after boiling out the reducing agent. The titanium was then found colorimetrically by hydrogen peroxide.

The combined alkaline filtrates were heated with excess of ammonium nitrate to separate alumina from most of the vanadium. The precipitate was redissolved in nitric acid, reprecipitated by ammonia, then ignited and weighed and redissolved by acid potassium sulphate. The sulphuric solution was reduced by hydrogen sulphide, boiled and filtered from platinum sulphide, again boiled in a current of carbon dioxide, and titrated hot with permanganate. Sulphur dioxide gas was then introduced, boiled out in a current of carbon dioxide, and the titration repeated, this second result being taken as representing the $V_2O_4$ equivalent to the $V_2O_5$ that still contaminated the $Al_2O_3$. Deducting the $V_2O_5$ thus found from the combined weight of $Al_2O_3$ and $V_2O_5$ gave the $Al_2O_3$ of the soluble constituent of the sandstone.

The combined filtrates from the alumina were evaporated and ignited to remove ammoniacal salts; the residue was converted to sulphates and united with the earlier solution of vanadium sulphate. The vanadium in it was then determined precisely in the manner already described, and when added to that found with the alumina and calculated to $V_2O_3$ gave the total in the sandstone.

A check was made by dissolving the sandstone in hydrofluoric and sulphuric acids in a current of carbon dioxide and titrating the $V_2O_3$ with permanganate. The figure thus obtained was a trifle below that found as above detailed, and this is to be attributed to the presence of a very little vanadium as $V_2O_5$ in carnotite, and probably as a lead vanadate. All the uranium and most of the trifling amount of lead could be extracted by cold dilute nitric acid, showing that they were foreign to the green substance.

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[a] Jour. Am. Chem. Soc., vol. 16, 1894, p. 578.

It is preferable to regard iron as in the ferric rather than the ferrous state; otherwise an equivalent amount of vanadium would have to be considered as $V_2O_5$, for which there is no occasion. It is quite possible that the iron is in part, if not wholly, foreign to the green matter, for it would be surprising to find an exposed sandstone free from ferric oxide.

If the iron is disregarded as probably extraneous, the ratios given below afford the following empirical formula:

$$H_{396}\ R'_{194}\ R''_{39}\ R'''_{610}\ Si_{763}\ O_{2775}.$$

$$\text{or, } H_{396}\ R'_{194}\ R''_{39}\ R'''_{610}\ (SiO_4)_{555}\ (Si_3O_8)_{69}.$$

Simplified, this is $H_{40}\ R'_{20}\ R''_4\ R'''_{61}\ X_{62.4}$.

In view of the fact that the mineral is probably not perfectly pure and of the uncertainties affecting the amount of water to be considered, etc., the above affords a close approach to the typical phengitic muscovite formula, $H_2KR'''_3X_3$, in which X comprises the groups $SiO_4$ and $Si_3O_8$.

| | Sandstone. | Ratios. | | Silicate. | Roscoelite. |
|---|---|---|---|---|---|
| Soluble in nitric acid: | | | | | |
| $SiO_2$ | 12.56 | 0.763 | | 46.06 | 45.17 |
| $TiO_2$ | .02 | | | | .78 |
| $V_2O_3$ | [a] 3.50 | .0846 | 0.305 | 12.84 | 24.01 |
| $Al_2O_3$ | 6.15 | .2206 | | 22.55 | 11.54 |
| $Fe_2O_3$ | .20 | .0046 | | .73 | FeO 1.60 |
| CaO | .12 | .0078 | .039 | .44 | |
| BaO | .37 | .0088 | | 1.35 | |
| MgO | .25 | .0228 | | .92 | 1.64 |
| $K_2O$ | 2.41 | .0938 | .097 | 8.84 | 10.37 |
| $Na_2O$ | .06 | .0035 | | .22 | .06 |
| $H_2O$ at 105° | .54 | .1100 | | 1.98 | .40 |
| $H_2O$ at 105°-300° | .14 | .0283 | | .51 | .17 |
| $H_2O$ above 300° | .97 | .1978 | .198 | 3.56 | 4.12 |
| $UO_3$ | .05 | | | | |
| PbO | .06 | | | | |
| $V_2O_5$ | .05 | | | | |
| Insoluble in nitric acid: | | | | | |
| Quartz, etc | [b] 72.24 | | | | |
| $H_2O$ at 105° | .04 | | | | |
| $H_2O$ above 105° | .20 | | | | |
| | 99.93 | | | 100.00 | 99.86 |

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[a] Oxidation by permanganate of the HFl and $H_2SO_4$ solution of the sandstone indicated 3.43 per cent $V_2O_3$ (mean of 3.50, 3.32, 3.48, 3.43).

[b] Containing about 65 per cent $SiO_2$, 7 per cent $Al_2O_3$ ($Fe_2O_3TiO_2$) and about 0.3 per cent of zircon *and other minerals* not decomposed by repeated evaporations with HFl and $H_2SO_4$.

*Traces of Li, Cu, Mo, Bi.* No Cl, $SO_3$, or $P_2O_5$.

An attempt to satisfactorily distribute the constituents among different chemical molecules would be futile, since, for instance, we do not know whether to ascribe the barium to a biotite or muscovite molecule, but the close approach to the muscovite ratio leaves little room to doubt the character of the body. Notwithstanding the chloritic aspect of the mineral under the microscope, the analysis shows that it can not be a chlorite, and we have undoubtedly to do with a body closely related to the mica roscoelite, wherein, however, the percentage proportions of $Al_2O_3$ and $V_2O_3$ are reversed, thus affording a further striking example of the mutual replaceability of these two oxides.

For comparison, the latest analysis of roscoelite from Placerville, Cal., is included in the above table. Peculiar, though unimportant, is the coincidence that the only two known localities for this mineral should bear the name Placerville.

## II. GREEN SANDSTONE COLORED BY CHROMIUM.

Other sandstones that were a much brighter green than the vanadiferous one were observed and collected by Doctor Ransome, both at Placerville and 60 miles distant in Sinbad Valley. The color suggested a salt of copper as its cause, but analysis showed it to be due to chromium. Time has not been found to determine the nature of this coloring body. It is very difficultly soluble, and thus presents greater hindrance to analysis than did the vanadium compound. It would be interesting to find that it is a micaceous mineral analogous to the one just described. Under the microscope it presents a chloritic appearance (Ransome). If opportunity offers, the problem of its nature may yet be attacked.

In still another greenish sandstone from the west bank of the Dolores River, near the mouth of La Sal Creek, analysis failed to show either vanadium or chromium.

## III. CARNOTITE ORES.

### OPTICAL EXAMINATION.

Dr. George P. Merrill has kindly submitted the following notes:

> The carnotite powder appears under the microscope in the form of exceedingly minute dust-like particles without crystal outlines and acting so faintly on polarized light as to at first seem almost amorphous. Much of the matter appears merely as a fine brownish clay, stained yellow by an amorphous pigment, but occasionally a well-defined fragment of a light-yellow translucent mineral is seen, which doubtless represents the vanadium compound in its condition of ideal purity.
>
> Working over a considerable amount of the powder I have found occasional clusters of this yellow mineral in the form of flattened radiating crystals with pyramidal terminations which are without evident pleochroism, polarize only in dull colors, and give extinctions always parallel to the axis of elongation. These are so minute (not over 0.25 mm. in length) and so thin that I have never been able to find a crystal so oriented as to give an opportunity of determining its exact character, and I can only say that the general shape is such as to suggest a hexagonal mineral, though this is by no means certain.

## CHEMICAL EXAMINATION OF CARNOTITE ORES.

The chemical problems involved in the analysis of the carnotite ores were peculiarly intricate. Ideal material was quite unobtainable and mechanical separation was impossible. The first specimen received happened to be of higher grade than any of the subsequent ones, containing about 5 per cent of sand grains and showing only very faint delicate reddish tracings, indicative of some foreign iron mineral. In mass it was of a beautiful canary-yellow color and easily broken down by pressure. Other specimens were more coherent, the degree depending altogether on the extent to which the sandstone had been impregnated and altered, and they sometimes showed more of the peculiar reddish admixture.

A fact only suspected in making the first analysis was confirmed by subsequent work on lower-grade ores, namely, that the vanadium existed in two conditions, in entirely distinct minerals, the greater part by far as pentavalent vanadium in the easily soluble carnotite, and a smaller, much less soluble portion, almost vanishing in the purest ores, in the trivalent state as a constituent of a silicate free from uranium. This observation explained the statement of Messrs. Poulot and Voillequé that they had found the low-grade ores relatively richer in vanadium, as compared with uranium, than the high-grade ores.

While the carnotite dissolves at once in cold dilute nitric acid, unfortunately the vanadiferous silicate is not quite insoluble, hence arose an important difficulty in the way of arriving at the true composition of the carnotite. It is true that in one case the silicate has been analyzed (p. 30), but it would be unsafe to correct the carnotite analysis on that basis. Moreover, the analyses were not all made on the same plan; some are less complete than others; they can not in some respects be rigidly compared with one another.

As to the carrying out of the analysis, various procedures were tried and no one was found which gave altogether satisfactory results as to each constituent, though many could be determined with the usual degree of accuracy. The difficulty was due mainly to vanadium and the small amount of phosphoric acid usually present. These two constituents were likely to be found in different precipitates and could not be separated from them at one stroke. Their complete removal from other bodies was at times impossible, and the weight of the latter had then to be corrected for these residual amounts.

Two entirely different lines of attack were open. One, that of Friedel and Cumenge, described in their paper on carnotite, had already been used with apparent success. It depends on rendering the vanadium insoluble in water by evaporating the nitric acid solution to dryness. Water extracts the alkalies and uranium without dissolving the vanadium, iron, or aluminum. Repetition of this process yielded Friedel and Cumenge satisfactory results. They recommended washing with water containing ammonium nitrate.

The other method involved the quantitative removal of vanadium by dry hydrochloric acid gas in the manner already described (pp. 18–19). Smith and Hibbs had shown that this method is perfect for alkaline pyrovanadates and it was hoped that it might succeed even with such complex mixtures as those under consideration. The distillations were made on the crude ore and also on the nitric acid solution after filtration from the insoluble matter and evaporation to thorough dryness. The action is immediate in the cold, copious red-brown vapors coming off and condensing in part as a dark red liquid in the tube. But repeated distillations and the application of heat were required to effect complete removal of the vanadium accompanied by the arsenic and molybdenum. After each distillation the contents of the boat had to be evaporated with nitric acid. The blue and green colorations apparent on adding this acid showed that the hydrochloric acid gas had reduced a good deal of the vanadium to a lower state of valence, and in this condition it was incapable of forming the volatile body.

A somewhat more satisfactory separation sometimes resulted when this distillation method was combined with that of Friedel and Cumenge by subjecting both the evaporated uranium nitrate solution and the residue insoluble in water to the action of hydrochloric acid gas.

The distillates obtained by either way were evaporated with sulphuric acid, the arsenic and molybdenum were separated by hydrogen sulphide, and the vanadium was then titrated by permanganate at a temperature near boiling and again after reduction by sulphur dioxide gas. In one or two cases when the temperature of distillation had been high and it was feared some iron had passed over, the distillates were evaporated in porcelain with nitric acid, transferred to a platinum crucible, evaporated therein with sulphuric acid, and fused with sodium carbonate. The aqueous extract was then treated as above for arsenic, molybdenum, and vanadium.

The mode of separation practiced by Friedel and Cumenge, while perhaps adequate for technical purposes in ores free from phosphorus,[a] does not in my hands give perfect satisfaction even then. It is impossible to prevent a little of the vanadium, also of the iron and aluminum, from going with the uranium, and on the other hand a little uranium may stay with the vanadium. Again when the vanadic acid is extracted from the residue by ammonia, as prescribed by Friedel and Cumenge, a little of the other constituents of the residue accompany the vanadium into solution.

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[a] With even only half a per cent of $P_2O_5$ many times that amount of $UO_3$ is rendered insoluble in water after evaporation to dryness with nitric acid. The compound formed is of a lemon-yellow color, which is masked by the separated vanadic acid until this has been removed by ammonia. This insoluble body was treated as follows in order to arrive at the $P_2O_5$ and $UO_3$ it contained. After solution in nitrite acid the phosphorus was precipitated by ammonium molybdate, and from the filtrate the uranium by three precipitations by ammonia.

If the combination of the two methods is used, it is better, after the vanadium has been removed by distillation, to treat the two residues separately at first instead of to unite them at once. Full details of this treatment are unnecessary and would unduly extend this paper. Suffice it to say that from the uranium portion the little iron and aluminum present are separated by ammonium sulphide and carbonate, and after removal of the latter and acidification the uranium can be thrown down by ammonium sulphide, and then by at least two precipitations by ammonia, or far better, by precipitating the neutral hydrochloric solution by freshly precipitated and alkali-free mercuric oxide at boiling heat, as prescribed by Alibegoff.[a] As pointed out by von Foullon and also by Alibegoff, contrary to certain still widely disseminated statements, ammonium sulphide does not afford a good separation of uranium from calcium. This is perhaps especially true if the solution contains any phosphorus. Again, contrary to another statement, it seems perfectly possible to separate uranium completely from alkalies by a few ammonia precipitations.

The finally ignited and weighed $U_3O_8$ was redissolved in nitric acid, filtered, if necessary ($SiO_2$, $Al_2O_3$, $Fe_2O_3$), and divided into two parts, one of which was tested for the very little $P_2O_5$ usually present, and the other for vanadium by conversion into sulphate, reduction by sulphur dioxide gas, and titration by very dilute permanganate solution.

### ANALYSES OF CARNOTITE ORES.

I. Copper Prince claim, Roc Creek, Montrose County, Colo.

According to the donor of the specimen, Mr. J. R. Duling, I-*a* is from the same lot as that from which Mr. Poulot obtained the specimens afterwards analyzed by Friedel and Cumenge.

II. Yellow Boy claim, La Sal Creek, Montrose County, Colo.

III. Yellow Bird claim, La Sal Creek, Montrose County, Colo.

The last two claims belong to the same group and the ore is from the same "blanket" (Voillequé).

I-*b* and II-*b*, the first analyses made, are not strictly comparable with the other analyses, since they represent the effect of warm dilute hydrochloric acid, whose greater action is shown by the nearly complete solution of the ferruginous admixture. Cold, dilute nitric acid was used for the other analyses.

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[a] Ann. Chem. u. Phar., vol. 233, 1886, p. 183; Zeit für anal. Chemie, vol. 26, 1887, p. 632.

| | I. | | | II. | | III. |
|---|---|---|---|---|---|---|
| | *a.* | *b.* | *c.* | *a.* | *b.* | |
| Insoluble | [a]7.10 | [b]8.34 | [c]19.00 | [d]10.33 | ([e]) | ([f]) |
| $UO_3$ | 54.89 | 52.25 | 47.42 | 54.00 | 52.28 | 20.51 |
| $V_2O_5$ | 18.49 | 18.35 | 15.76 | 18.05 | 17.50 | 7.20 |
| $P_2O_5$ | .80 | .35 | .40 | .05 | trace | none |
| $As_2O_5$ | trace | .25 | none | none | none | none |
| $Al_2O_3$ | .09 | (?) | .08 | .29 | (?) | .08? |
| $Fe_2O_3$ | .21 | 1.77 | .72 | .42 | 3.36 | .25 |
| CaO | 3.34 | 2.85 | 2.57 | 1.86 | 1.85 | 1.64 |
| SrO | .02 | (?) | (?) | trace | trace | (?) |
| BaO | .90 | .72 | .65 | 2.83 | 3.21 | .29 |
| MgO | .22 | .20 | .24 | .14 | .17 | .07 |
| $K_2O$ | 6.52 | 6.73 | 6.57 | 5.46 | 5.11 | 1.51 |
| $Na_2O$ | .14 | .09 | .07 | .13 | .02? | .01 |
| $Li_2O$ | trace | (?) | (?) | trace | (?) | (?) |
| $H_2O$ at 105° | 2.43 | 2.59 | 1.85 | 3.16 | 4.52 | 1.85 |
| $H_2O$ at 350° | 2.11 | 3.06 | 2.79 | 2.21 | 3.49 | [g]1.64 |
| $H_2O$ above 350° | none | none | none | none | .......... | [h].19 |
| PbO | .13 | .25 | .18 | .07 | .......... | .09 |
| CuO | .15 | .20 | .22 | trace | .......... | trace |
| $SO_3$ | none | .12 | .18 | none | .......... | none |
| $MoO_3$ | .18 | .23 | .18 | .05 | .......... | .04 |
| $SiO_2$ | .15 | .06 | .13 | .20 | .......... | .07 |
| $TiO_2$ | .03 | .10 | (?) | (?) | .......... | .06? |
| $CO_2$ | .56 | .33 | none | none | .......... | ........ |
| | 98.46 | 98.84 | 99.01 | 99.25 | .......... | ........ |

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[a] Containing 0.54 $H_2O$, 0.09 $V_2O_3$.

[b] The insoluble matter had the following composition: $SiO_2$, 5.18; $V_2O_3$, 0.21; $P_2O_5$, 0.09; $K_2O$, 0.26; $Na_2O$, 0.04; $H_2O$, 105°, 0.56, 350°, 0.32, +350°, 0.48; $Al_2O_3$, $TiO_2$, $Cr_2O_3$, etc., by diff., 1.20. The $SO_3$ of this ore is not combined with BaO, for it is wholly extracted by dilute acids. Once in solution the acid used is sufficient to prevent its immediate precipitation as $BaSO_4$ by union with some of the barium present. The same holds true for the $SO_3$ of I–*c*.

[c] This material was obtained by floating off the finer matter, allowing it to settle, collecting on a Gooch filter, and drying it in a current of air drawn through the crucible. The insoluble matter held in addition to 16.41 quartz and silicates (including 0.39 $V_2O_3$ and a little $UO_3$), $H_2O$, 105°, 0.83; 300°, 0.73; +300°, 1.03; total $H_2O$, 2.59.

[d] Containing 0.16 $V_2O_3$ and 1.90 $H_2O$.

[e] The insoluble matter contained besides quartz and silicates: 0.25 $V_2O_3$, 0.21 $K_2O$, 0.05 $Na_2O$.

[f] This analysis was made purposely on a relatively poor ore, furnished by Messrs. Poulot and Voillequé, with the object of determining, if possible, the composition of the vanadiferous silicate which it contained (see p. 30). The data for calculating the $H_2O$ values of both analyses are as follows:

| | Ore. | After extraction of carnotite by cold, dilute nitric acid. | After extraction of residue by hot nitric acid, sodium carbonate, etc. |
|---|---|---|---|
| | (*a*) | (*b*) | (*c*) |
| $H_2O$ at 105° | 3.53 | 1.68 | 0.02 |
| $H_2O$ at 300° | 2.11 | .47 | .03 |
| $H_2O$ above 300° | .83 | .64 | .02 |
| | 6.47 | 2.79 | .07 |

*a–b* furnishes the values for the carnotite as shown in analysis III; *b–c* gives those for the less soluble silicate (p. 30).

[g] At 300°.

[h] Above 300°.

## DISCUSSION OF THE CARNOTITE ANALYSES.

It will be noted that a somewhat marked deficiency appears in most of the analyses, the cause of which is quite unknown. Great care was exercised in most cases and especially in those which show the greatest loss. It seems hardly possible that any serious constant loss of a known constituent should have occurred, but the only alternative demands the presence of an element or elements unnoticed and which can not have been weighed with the known constituents. The researches of M. and Mme. Curie have shown that these ores contain traces of radio-active elements, precipitated the one by hydrogen sulphide, the other by sulphuric acid. Their presence, however, in quantity sufficient to account for the observed losses in the above analyses, especially when 10 grams of ore were operated on, could not possibly have escaped observation. To whatever cause it may be due, this loss alone suffices to render somewhat uncertain any calculations based on the analytical figures, though if the loss is to be ascribed to uranium or vanadium the ratios would not be sufficiently affected to obscure any simple relations that might exist.[a]

Another difficulty is the impossibility of knowing what constituents to exclude and what to include in deducing molecular ratios. It is certain that most if not all of the iron is foreign to the yellow body. It is probable that phosphorus is likewise so, since its extraction by dilute acids does not keep pace with that of the uranium and vanadium. It may possibly be in combination with the iron, in part at least. The alumina doubtless is derived from the vanadiferous silicate which seems to exist in all the ores and which is not quite insoluble in cold dilute acids. If so, a small portion of the vanadium, potassium, magnesium, and water are to be attributed to this mineral, but a general correction based on the analysis of this compound (see p. 30) would not be justifiable. Its application leads to nothing definite, even in the case of the particular ore No. III, from which the silicate analyzed was derived.

In the following tables are given first the recalculated analyses and then the molecular ratios. All those constituents have been excluded which pretty certainly do not belong to the carnotite, but small portions of some of those retained are unquestionably extraneous. In two cases (I-*a* and I-*b*) a certain proportion of lime has been arbitrarily excluded equivalent to the acid anhydrides $CO_2$, $MoO_3$, and $SO_3$, less what is needed to offset PbO and CuO.

---

[a] According to Dr. Harry C. Jones, of Johns Hopkins University, who very kindly undertook to examine a specimen of the Copper Prince ore for rare gaseous elements, helium is not present. Faint hydrogen lines were observed, the source of which was ascribed to water vapor. Other lines, due probably to hydrocarbons, were fairly strong, but the specimen had been long enough exposed in our laboratory and elsewhere to have accumulated enough dust to account for them.

CARNOTITE ANALYSES RECALCULATED.

| | I-*a*. | I-*b*. | I-*c*. | II-*a*. | II-*b*. | III. |
|---|---|---|---|---|---|---|
| $V_2O_5$ | 20.72 | 21.09 | 20.12 | 20.54 | 19.85 | 20.62 |
| $P_2O_5$ | .90 | .40 | .51 | .06 | ........ | ........ |
| $As_2O_5$ | ........ | .29 | ........ | ........ | ........ | ........ |
| $UO_3$ | 61.53 | 60.06 | 60.55 | 61.44 | 59.31 | 58.75 |
| CaO | 3.03 | 2.77 | 3.28 | 2.11 | 2.10 | 4.70 |
| BaO(SrO) | 1.03 | .83 | .83 | 3.22 | 3.64 | .83 |
| MgO | .25 | .23 | .31 | .16 | .19 | .20 |
| $K_2O$ | 7.31 | 7.73 | 8.39 | 6.21 | 5.80 | 4.33 |
| $Na_2O$ | .15 | .10 | .09 | .15 | .02 | .03 |
| $H_2O-105°$ | 2.72 | 2.98 | 2.36 | 3.59 | 5.13 | 5.30 |
| $H_2O+105°$ | 2.36 | 3.52 | 3.56 | 2.52 | 3.96 | 5.24 |
| | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 |

MOLECULAR RATIOS.

| | I-*a*. | | I-*b*. | | I-*c*. | | II-*a*. | | II-*b*. | | III. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| $V_2O_5$ | 1133 | 1196 | 1208 | 1248 | 1101 | 1137 | 1124 | 1128 | | 1081 | | 1128 |
| $P_2O_5$ | 63 | | 28 | | 36 | | 4 | | | ........ | | ........ |
| $As_2O_5$ | ..... | | 12 | | .... | | .... | | | ........ | | ........ |
| $UO_3$ | | 2140 | | 2088 | | 2105 | | 2136 | | 2062 | | 2043 |
| CaO | 541 | 671 | 495 | 606 | 586 | 718 | 377 | 627 | 375 | 660 | 839 | 943 |
| BaO(SrO) | 68 | | 54 | | 54 | | 210 | | 238 | | 54 | |
| MgO | 62 | | 57 | | 78 | | 40 | | 47 | | 50 | |
| $K_2O$ | 776 | 809 | 826 | 848 | 891 | 911 | 659 | 692 | 616 | 620 | 460 | 466 |
| $Na_2O$ | 33 | | 22 | | 20 | | 33 | | 4 | | 6 | |
| $H_2O-105°$ | | 1511 | | 1656 | | 1311 | | 1994 | | 2850 | | 2944 |
| $H_2O+105°$ | | 1311 | | 1956 | | 1978 | | 1400 | | 2200 | | 2911 |

These ratios lead to the following empirical formulas, in which only the water given off above 105° is considered. Since the water is wholly removable below 350° it is regarded as water of crystallization and not of constitution.

| | R′. | R″. | Rᵛ. | U. | O. | $H_2O$. |
|---|---|---|---|---|---|---|
| I-*a* | 1618 | 671 | 2392 | 2140 | 13880 | 1311 |
| I-*b* | 1696 | 606 | 2496 | 2088 | 13958 | 1956 |
| I-*c* | 1822 | 718 | 2274 | 2105 | 13629 | 1978 |
| II-*a* | 1384 | 627 | 2256 | 2136 | 13367 | 1400 |
| II-*b* | 1240 | 660 | 2162 | 2062 | 12871 | 2200 |
| III | 932 | 943 | 2256 | 2043 | 13178 | 2911 |

If it be assumed that the bivalent elements offer the most accurate determinations, the above ratios may be reduced to the following simpler terms on that basis:

|  | $R'$. | $R''$. | $R^v$. | U. | O. | $H_2O$. |
|---|---|---|---|---|---|---|
| I-*a* | 2.41 | 1 | 3.56 | 3.19 | 20.68 | 1.95 |
| I-*b* | 2.80 | 1 | 4.12 | 3.45 | 23.03 | 3.23 |
| I-*c* | 2.54 | 1 | 3.17 | 2.93 | 18.98 | 2.75 |
| II-*a* | 2.21 | 1 | 3.60 | 3.41 | 21.32 | 2.23 |
| II-*b* | 1.88 | 1 | 3.27 | 3.12 | 19.50 | 3.33 |
| III | .99 | 1 | 2.39 | 2.16 | 13.98 | 3.09 |

The results, however, show a great lack of agreement and wide variation. It is plain that no probable formula can be calculated for the yellow body. The variations are of such a nature as to indicate in the plainest manner that it is a mixture of several substances.

So detailed a discussion as the foregoing would hardly have been justified, in view of the negative conclusions arrived at, but for the fact that Messrs. Friedel and Cumenge in their paper announced a simple formula for the body examined by them and gave it the specific name carnotite. Their published analyses are as given below, from which they have excluded considerable sand and traces of barium, aluminum, lead, copper, and radio-active bodies as present in excessively small quantities. They make no mention of calcium and admit that their values for water are open to doubt. The formula deduced by them is $2U_2O_3$,[a] $V_2O_5$, $K_2O$, $3H_2O$.

|  | Found. |  |  |  | Calculated. |
|---|---|---|---|---|---|
|  | 1. | 2. | 3. | 4. |  |
| $U_2O_3$[a] | 64.70 | 62.46 |  |  | 63.54 |
| $V_2O_5$ | 20.31 | 19.95 |  |  | 20.12 |
| $K_2O$ | 10.97 | 11.09 |  |  | 10.37 |
| $H_2O$ |  |  | 5.29 | 4.81 | 5.95 |
| $Fe_2O_3$ | .96 | .65 |  |  |  |
|  |  |  |  |  | 99.98 |

[a] Old notation, equivalent to the modern $UO_3$.

It appears from these analyses that Messrs. Friedel and Cumenge by great good fortune obtained a variety of samples of the pure potassium compound, free from calcium and without appreciable admixture of barium. This is very remarkable in view of the fact that all the ores from different localities, examined by me show large admixture of calcium or barium salts or both, even that which is certified to have come from the same lot as that from which their material was taken. The French authors give a brief outline of their methods of analysis. That one which afforded them the best results would involve the weighing of any calcium present as sulphate along with the potassium, on the assumption that the presence of that element had been overlooked. It is much to be desired that a reanalysis of their material should be made, if there is any of it still available, in order to clear up the doubt connected with the first analysis.

In the light of the evidence herein set forth, the existence of a distinct mineral species having the composition claimed for carnotite can by no means be considered as established.

### AVERAGE QUALITY OF MARKETED ORE.

As these carnotite ore bodies are being exploited for the market, it is of some interest to know the average quality of each commercial lot. A carefully prepared sample representing several tons of ore was received from one of the commercial houses of Denver and was found to carry 11.49 per cent of uranium counted as $U_3O_8$, and 6.40 per cent of vanadium counted as $V_2O_5$. Over one-sixth of the vanadium existed, however, in the trivalent state, not as a constituent of the yellow body, but doubtless of a silicate like the one whose composition is given on page 30.

### COMMERCIAL ASSAY.

The commercial assay of these ores has presented difficulties to the technical chemist, the results being sometimes very discordant.

As to uranium, this is not surprising. The methods that have probably been commonly employed will give varying results, according to the contents of the ore in phosphorus and alkaline earths. Possibly the old Patera process, described in most text-books on analytical chemistry, might be made to serve, with modifications called for by the large amount of vanadium present.

The assay for vanadium presents little difficulty and does not require much time. The ore is fused with sodium carbonate, leached with water, and the fusion repeated on the residue. The combined filtrates are acidified by sulphuric acid, arsenic and molybdenum are precipitated in the hot solution by hydrogen sulphide, whereby the $V_2O_5$ is reduced to $V_2O_4$. After filtration and expulsion of hydrogen sulphide by boiling, the vanadium is titrated in hot solution by permanganate. It is then reduced by sulphur dioxide gas, and after boiling this out the titration is repeated. The results are exact, and they are not affected by the uranium that may be present.

## IV. COMPOSITION OF VANADIFEROUS SILICATE IN CARNOTITE ORE.

As already mentioned, the ores contain a vanadiferous silicate free from uranium. To the end of ascertaining its composition, if possible, ore No. III (p. 25), from the Yellow Bird claim, low in carnotite, but relatively rich in vanadium, was treated as follows:

The carnotite from 10 grams was extracted by cold, dilute nitric acid, and the well-washed residue, consisting of coarse sand and an utterly amorphous mud, by 4 per cent sodium carbonate solution to get rid of the small amount of silica presumably set free, but not dissolved by the acid. This amounted to 0.35 per cent in duplicate determinations, and together with 0.06 per cent in the acid solution, or 0.41 per cent in all, may serve as a maximum figure by which to gage the action of the cold acid on the silicate or silicates in the ore. The residue was then digested for several hours with warm nitric acid of about 1.2 specific gravity until, as shown by a companion test, its action had ceased. It was then filtered, washed, and digested with 5 per cent sodium carbonate solution to dissolve the copious deposit of silica. The final residue was collected in a Gooch crucible, washed with sodium carbonate, followed by dilute nitric acid to remove all alkali, then by alcohol to prevent turbidity in the filtrate, and dried by suction of the pump. In it the water was determined at different temperatures; also its general composition.[a] From the sodium carbonate filtrate the silica was obtained by two evaporations and filtrations; also the trifling amount held by the acid solution. This latter was then fully analyzed and the complete results follow:

| | Per cent in ore. | Per cent calculated to 100. | Ratios. | |
|---|---|---|---|---|
| $SiO_2$ | 6.48 | 43.94 | | 0.7275 |
| $Al_2O_3$ | [b] 2.445 | 16.58 | 0.1622 | .2426 |
| $V_2O_3$ | [c] .965 | 6.54 | .0434 | |
| $Fe_2O_3$ | [d] .875 | 5.93 | .0370 | |
| CaO | .035 | .24 | .0043 | .1142 |
| MgO | [e] .654 | 4.43 | .1099 | |
| $K_2O$ | .546 | 3.70 | .0393 | .0430 |
| $Na_2O$ | .03 | .20 | .0037 | |
| $H_2O$ at 105° C | [f] 1.66 | 11.26 | | .9222 |
| $H_2O$ at 300° C | [f] .44 | 2.98 | | .1655 |
| $H_2O$ above 300° C | [f] .62 | 4.20 | | .2333 |
| | 14.75 | 100.00 | ............. | |

[a] $SiO_2$, 48.89; $Al_2O_3$, $Fe_2O_3$, zircon, etc., 0.44; $TiO_2$, 0.08, MgO, 0.01; $K_2O$, 0.11; $Na_2O$, 0.03; $H_2O$, 0.07; total, 49.63. [b] 2.45 and 2.44. [c] 0.93 and 1.00. [d] 0.87 and 0.88. [e] 0.652 and 0.657. [f] Determined *on a separate portion of the* same powdered sample. See p. 25, footnote, for data.

*Also traces of titanium,* manganese, and lithium.

On the improbable assumption that the iron is to be wholly included, and regarding only the water given off above 300°, the following ratios result:

$$H_{467}\ R'_{86}\ R''_{114}\ R'''_{485}\ Si_{727}\ O_{2573};$$

which become, if the iron is excluded,

$$H_{467}\ R'_{86}\ R''_{114}\ R'''_{411}\ Si_{727}\ O_{2462}.$$

These figures, while strongly suggesting definite ratios between certain of the constituents, do not under the circumstances warrant the deduction of a formula, nor do they lead to the same conclusion as in the case of the green cementing material of the sandstone at Placerville. The phengite-muscovite ratio of that is not apparent here; yet it is not at all unlikely that a mineral like the one from Placerville is present, but contaminated with some other. In fact it would be surprising to find anything but a mixture in sandstones so thoroughly altered. The very existence of such vanadiferous transition products is itself highly interesting, and these tedious analyses were not therefore made in vain. The mud-like amorphous character of this material precludes any hope of aid from the microscope in solving the question of its homogeneity.

## SUMMARY.

The body called carnotite is probably a mixture of minerals, the exact nature of which analysis fails to reveal. Instead of being the pure uranyl-potassium vanadate, it is to a large extent made up of calcium and barium compounds. Intimately mixed with and entirely obscured by it is an amorphous substance—a silicate or mixture of silicates— containing vanadium in the trivalent state probably replacing aluminum.

The deposits of carnotite, though distributed over a wide area of country, are for the most part, if not altogether, very superficial in character and of recent origin.

The green coloring and cementing material of certain sandstones near Placerville, Colo., is a cryptocrystalline alumino-vanadio-potassium silicate resembling roscoelite, but with the percentage proportions of $Al_2O_3$ and $V_2O_3$ reversed. It constitutes over 25 per cent of portions of the sandstone, and contains nearly 13 per cent of $V_2O_3$, the latter amounting in the maximum case observed to 3.5 per cent of the sandstone.

As yet (1900) these highly vanadiferous sandstones have been found only at Placerville, where it is intended to work them for vanadium. Carnotite is associated with them in only trifling amount.

Other sandstones noticed owe their bright green color to chromium.

In yet another case where the color was dull green this was not due to either chromium or vanadium.

# SOME ADDITIONS TO THE ALUNITE-JAROSITE GROUP OF MINERALS.

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By W. F. Hillebrand and S. L. Penfield.

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Two new varieties of jarosite will be described in the present paper. One is from Nevada, and was collected by Mr. H. W. Turner, of the United States Geological Survey, and sent to the Survey laboratory at Washington for identification; the other is from New Mexico, and was sent by Mr. J. H. Porter, of Denver, Colo., to the mineralogical laboratory of the Sheffield Scientific School. Except for slight differences in color the two minerals look exactly alike, each consisting of minute, isolated, tabular crystals, which, as may be seen with the microscope, are composed of combinations of a rhombohedron with largely developed basal planes. By chance it happened that the present writers discovered that they were both engaged in the investigation of compounds belonging evidently to the same group, and it was decided to bring the results together into one paper.

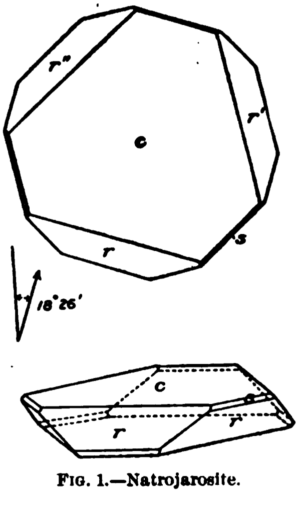

FIG. 1.—Natrojarosite.

## NATROJAROSITE.

The material collected by Mr. Turner was obtained on the east side of Soda Springs Valley, Nevada, on the road from Sodaville to the Vulcan copper mine. It consists of a glistening powder, made up of perfect crystals having the habit shown in fig. 1, although generally only one rhombohedron, *r*, is present instead of two, as shown in the figure. The largest crystals observed were 0.15 mm. wide and 0.025 mm. thick, and the general average would not be over half that size. In spite of their minuteness, however, it was possible to measure the angles of the crystals with the reflection goniometer, the chief difficulty arising not so much from their small size as from the vicinal character of the basal planes. After repeated trials a crystal was found having a fairly good basal plane, and from this crystal the following angles were obtained:

| | Measured. | | Measured. | Calculated. |
|---|---|---|---|---|
| $c \wedge r$, | $0001 \wedge 10\bar{1}1 = 51° 53'$ * | $c \wedge s$, | $0001 \wedge 02\bar{2}1 = 68° 42'$ | 68° 35′ |
| $c \wedge r'$, | $0001 \wedge \bar{1}101 = 51\ \ 53$ | $c \wedge s'$, | $0001 \wedge \bar{2}021 = 68\ \ 48$ | 68 35 |
| $c \wedge r''$, | $0001 \wedge 0\bar{1}11 = 52\ \ 26$ | $r \wedge r'$, | $10\bar{1}1 \wedge \bar{1}101 = 86\ \ 5$ | 85 54 |

The crystals belonged to the rhombohedral division of the hexagonal system, and the angle $c \wedge r$, 51° 53′, which is probably very nearly correct, has been assumed as fundamental, and from it the following axial ratio has been calculated:

$$c = 1.104.$$

That the axial ratio as given is very near the truth is shown by the fact that the measurements of $c \wedge s$ and $r \wedge r$ do not vary many minutes from the calculated values; while on a number of other crystals, measurement of the angle $c \wedge r$, though varying considerably, was found to be not far from 52°. The angles of $c \wedge r$ and $r \wedge r'$ of the ordinary potassium jarosite are 55° 16′ and 90° 45′, respectively.

Under the microscope the crystals exhibit normal optical properties. With a high-power lens and convergent light the thicker crystals show the dark cross and the beginnings of the first ring of the interference figure. The birefringence is negative. The color of single crystals, when seen under the microscope in transmitted light, is golden yellow. Many of the crystals show numerous brown inclusions. The color shown by a mass of the crystals is yellowish brown, and the material glistens, owing to reflections from the basal planes of the minute crystals.

The material used for the chemical analysis was the purest that could be obtained, although crystals containing the brownish inclusions just mentioned could not be avoided, and there were occasional brown ferruginous particles mixed with the crystals. The specific gravity of the material was found to be 3.18 at 30.5° C. The results of the analysis by Hillebrand are as follows:

| | | Ratio. | |
|---|---|---|---|
| $Fe_2O_3$ | 50.98 | 0.319 | 3.29 |
| $Na_2O$ [a] | 6.03 | .094 | 1.01 |
| $K_2O$ | .35 | .004 | |
| $SO_3$ | 30.96 | .387 | 4.00 |
| $H_2O$ below 105° | .12 | | |
| $H_2O$ above 105° | 11.03 | .613 | 6.33 |
| $As_2O_5$ | .20 | | |
| $SiO_2$ | .23 | | |
| $CaO$ | .04 | | |
| | 99.94 | | |

---

[a] Of the soda 0.22 per cent is not extracted by hot water after full ignition of the mineral, and hence may belong to a feldspar or some other foreign mineral. Only 5.81 per cent is assumed to belong to the jarosite and used in deriving the molecular value.

The ratio of $Fe_2O_3:Na_2O:SO_3:H_2O$ is evidently 3:1:4:6 as in ordinary jarosite, where the alkali is potash instead of soda. The slight excess of $Fe_2O_3$ and $H_2O$, as indicated by the ratio, is evidently due to some ferric hydroxide; probably the dark ferruginous impurities seen under the microscope are in part responsible for this, and there are also traces of some arsenate and silicate present. By attributing the excess of $Fe_2O_3$ and $H_2O$ to impurities, it is found that 94 per cent of the material analyzed may be regarded as pure natrojarosite, as indicated below:

| | After deducting impurities. | Theory for $Na_2Fe_6[OH]_{12}[SO_4]_4$. |
|---|---|---|
| $Fe_2O_3$ | 46.43 or 49.39 | 49.49 |
| $Na_2O$ | 5.81 or 6.18 | 6.39 |
| $K_2O$ | .35 or .37 | ........... |
| $SO_3$ | 30.96 or 32.94 | 32.99 |
| $H_2O$ | 10.45 or 11.12 | 11.13 |
| | 94.00 or 100.00 | 100.00 |

That 6 per cent of impurities should be present in a crystalline powder such as was analyzed is not surprising when it is taken into consideration that it would require something like 2,500,000 crystals to make one gram of material, the estimation being based on the assumption that the crystals are 0.10 mm. in axial diameter and 0.02 mm. thick, which is certainly above their average size.

Among the specimens from Cooks Peak, New Mexico, sent to the Sheffield laboratory by Mr. Porter, were some masses of a rather firmly cemented aggregate of minute crystals of a mineral of the jarosite group. The specimens are of a brownish-yellow color, and have in places the glistening appearance of a mica-schist. They also look as though they had been subjected to pressure and had been somewhat sheared. The material is rather easily crushed, and the powder, when examined with the microscope, exhibits the proprieties of the natrojarosite just described. The crystals are associated with a little limonite and quartz, and pure material for analysis could not be obtained. Only a partial analysis, therefore, was undertaken with the following results:

| | |
|---|---|
| $Fe_2O_3$ | 55.60 |
| $Na_2O$ | 4.49 |
| $K_2O$ | .77 |
| PbO | .96 |

$SO_3$ and $H_2O$ were present but not determined. The results are sufficient to indicate that the material is essentially natrojarosite.

## PLUMBOJAROSITE.

This material is from Cooks Peak, New Mexico. It occurs as a glistening, crystalline powder and as loosely cohering masses which may easily be crushed by pressure between the fingers. The crystals are very symmetrical, and are exactly like those of natrojarosite (fig. 1), although generally only one rhombohedron, *r*, is present. On the average the crystals are a trifle smaller and noticeably thinner than those of natrojarosite. A number of crystals were measured on the reflecting goniometer, the chief difficulty arising rather from the vicinal character of the faces than from their small size. One unusually large crystal, 0.28 mm. broad and 0.015 mm. thick, was finally found, having the development shown in fig. 2, which is unusual, for generally *r* ($10\bar{1}1$) and not *s* ($02\bar{2}1$) is the prevailing rhombohedron. Fortunately the crystal was so taken up on a minute point of wax that the measurement of $s \wedge s$ in three rhombohedral zones was possible. The results of five measurements of *s* *s* over the upper and lower pole edges varied between 109° 5′ and 109° 30′, the average being 109° 16′; while six measurements over the middle edges varied between 70° 10′ and 71° 00′, the average being 70° 36′. The average of the two supplementary values gives $s \wedge s$, $2\bar{2}01 \wedge 02\bar{2}1 = 109°\ 20'$, which has been assumed as fundamental, and from it the following axial ratio has been calculated:

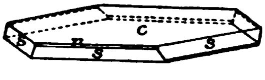

FIG. 2.—Plumbojarosite.

$$c = 1.216.$$

On the crystal from which the foregoing measurements were obtained the basal plane was vicinal and hence no reliable measurements of $c \wedge s$ could be had from it. On a number of other crystals, however, the angle of $c \wedge r$ was measured with varying results, the variation resulting from the uncertainty of the reflections from the basal planes. Four measurements of $c \wedge r$, which were recorded in the notebook as derived from the best reflections, varied between 54° 15′ and 54° 44′, the average being 54° 30′, while $c \wedge r$, $0001 \wedge 10\bar{1}1$, by calculation from the fundamental measurement, is 54° 32′. Hence it may be assumed that the axial ratio as established is reasonably exact. The calculated value of $r \wedge r$, $10\bar{1}1 \wedge \bar{1}101$, is 89° 42′.

In polarized light the crystals exhibit normal optical properties and negative birefringence. Being on the average thinner than crystals of natrojarosite, it is seldom that, with the highest powers and convergent light, even the beginning of the first ring of the uniaxial interference figure is visible. Individual crystals show under the microscope in transmitted light a golden-yellow color. A mass of crystals has the appearance of a glistening dark-brown powder, the color being decidedly darker than that of natrojarosite.

The analysis of the mineral was made on the very best material, having a specific gravity of 3.665 at 30° C. The results are surprising, and were wholly unlooked for, since it is found that this jarosite contains lead in the place of alkalies. The results by Hillebrand are as follows:

| | I. | II. | III. | IV. | Mean. | Ratio. | |
|---|---|---|---|---|---|---|---|
| $Fe_2O_3$ | 42.36 | 42.38 | | | 42.37 | 0.265 | 3.15 |
| $Al_2O_3$?[a] | .12 | .08 | 0.11 | | .10 | .001 | |
| PbO | 19.69 | 19.99 | 19.89 | 19.79 | 19.84 | .089 | 1.05 |
| $K_2O$ | .17 | | | | .17 | | |
| $Na_2O$ | [b].21 | | | | [b].21 | | |
| $SO_3$ | 27.05 | 27.07 | | | 27.06 | .338 | 4.00 |
| $H_2O$ below 105° | .02 | | | | .02 | | |
| $H_2O$ above 105° | 9.59 | 9.49 | | | 9.54 | .530 | 6.27 |
| $SiO_2$ | .56 | .51 | .47 | | .51 | | |
| CuO | .27 | .27 | | | .27 | | |
| CaO | .05 | | | | .05 | | |
| MgO | .01 | | | | .01 | | |
| | | | | | 100.15 | | |

[a] The presence of alumina was not definitely proved. The figures here given are the differences between the several weights of the ammonia precipitates and those of the ferric iron in them, as determined by permanganate after reduction by hydrogen sulphide.

[b] Probably somewhat high.

The ratio of $Fe_2O_3 : PbO : SO_3 : H_2O$ is very close to 3 : 1 : 4 : 6, indicating that the mineral is a variety of jarosite, and the slight excess of $Fe_2O_3$, $H_2O$, and PbO + alkalies may be accounted for by assuming that slight impurities are present, partly ferric hydroxide, in part some lead salt, and perhaps a soluble silicate, as shown by the complete solubility of the silica in acids. Assuming that the ratio is exactly 3 : 1 : 4 : 6, it is found that 4.36 per cent of impurities are present, and the remaining 95.64 per cent may then be regarded as plumbojarosite, as follows:

| | | Theory for $PbFe_6[OH]_{12}[SO_4]_4$. |
|---|---|---|
| $Fe_2O_3$ | 40.59 or | 42.44 |
| PbO | 18.86 or | 19.72 |
| $SO_3$ | 27.06 or | 28.29 |
| $H_2O$ | 9.13 or | 9.55 |
| | 95.64 or | 100.00 |

Since it took probably 2,500,000 crystals of natrojarosite to make 1 gram of material, it certainly must have taken fully 4,000,000 to make a gram of plumbojarosite, for the crystals of the latter mineral, though somewhat heavier, are decidedly thinner than those of the former; hence the presence of 4.5 per cent of impurities in such a *crystalline product* is not to be wondered at.

## JAROSITE AND ALUNITE.

As may be seen from the published analyses of these minerals, the alkali metal they contain is almost always potassium, though sodium is at times present. The formulas assigned to the two minerals are therefore $K_2O+3Fe_2O_3+4SO_3+6H_2O$ and $K_2O+3Al_2O_3+4SO_3+6H_2O$, which may be variously expressed, as will be indicated later.

A mineral corresponding to natrojarosite of this article, though containing a little potash, has been described by W. P. Headden[a] from the Buxton mine, Lawrence County, S. Dak. The crystals are described as scales, consisting of a combination of base and rhombohedron. The material analyzed was evidently somewhat impure, as quartz and some $As_2O_5$ are reported. As the $As_2O_5$ evidently does not belong to jarosite, the assumption may be made that some scorodite, $FeAsO_4.2H_2O$, is present, and the results of Headden's analysis may then be interpreted as follows:

| | Original analysis. | Scorodite and quartz. | Natrojarosite. | Ratio. |
|---|---|---|---|---|
| $Fe_2O_3$ | 46.27 | 1.60 | 44.67 or 50.10 | 3.13 |
| $Na_2O$ | 4.35 | .......... | 4.35 or 4.86 | 1.04 |
| $K_2O$ | 1.47 | .......... | 1.47 or 1.65 | |
| CaO | .39 | .......... | .39 or .44 | |
| $SO_3$ | 28.46 | .......... | 28.46 or 31.93 | 4.00 |
| $H_2O$ | 10.55 | .72 | 9.83 or 11.02 | 6.13 |
| $As_2O_5$ | 2.36 | 2.36 | .................... | ....... |
| Quartz | 6.10 | 6.10 | .................... | ....... |
| | 99.95 | 10.78 | 89.17 or 100.00 | ....... |

Thus, assuming the presence of 4.68 per cent of scorodite and 6.10 of quartz, and deducting them, the remainder agrees very closely with natrojarosite, giving a good ratio, very near 3:1:4:6.

Alunite containing considerable soda has been described by Whitman Cross[b] from Rosita Hills, Colorado, and by E. B. Hurlburt[c] from Red Mountain, Colorado, and analyses of both minerals show about equal percentages of $K_2O$ and $Na_2O$, or a molecular ratio of $K_2O:Na_2O=4:7$. The occurrence, therefore, of sodium in the jarosite-alunite group is in accordance with previous observations, but the case is quite different with lead. As far as the present writers are aware, this is the first instance on record where lead has been observed isomorphous with the alkali metals. It is interesting to note that the alunite from Red Mountain, Colorado, occurs as a crystalline powder, the crystals being exactly like those of natrojarosite and plumbojarosite, except that they are a trifle smaller and white, or colorless, when seen under the microscope.

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[a] *Am. Jour. Sci., 3d ser., 46, 1893, p. 24.*

[b] Ibid., 41, 1891, p. 472.

[c] Ibid., 48, 1894, p. 130.

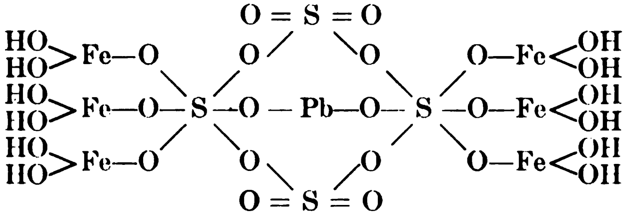
From a chemical standpoint the most interesting feature of the new minerals is the light they throw upon the isomorphism of potassium, sodium, and lead. Ordinarily, even potassium and sodium are not isomorphous, as shown by the fact that their simple salts seldom crystallize in the same form. Although KCl and NaCl both crystallize in cubes, it is not certain that both salts belong to the same group of the isometric system. It has been shown, for example, by etching, that KCl crystallizes like $NH_4Cl$ in the plagihedral group of the isometric system, while the etchings produced on halite seem to indicate that it crystallizes in the normal group. Again, at Stassfurt, Germany, sylvite and halite both occur crystallized side by side upon the same hand specimen, instead of mixing as isomorphous molecules. Even in such complex molecular compounds as the feldspars, the potassium and sodium salts crystallize as orthoclase and albite, rather than as isomorphous mixtures. Lastly, potassium has a strong tendency to form alums, which is not shared by sodium. In contrast to these differences in chemical nature, we have in the jarosite-alunite group of minerals not only the alkali-metals, potassium and sodium, but, what seems still more remarkable, *lead*, playing the same rôle in the compounds, and yielding crystals which are surprisingly alike in all their physical properties. The writers can at present offer no other reason for the isomorphism in the group of minerals under consideration than that the alkalies and lead play so small a rôle, and the remaining constituents so prominent a part in the complex chemical molecules, that the latter control or dominate the crystallization by virtue of what may be called their *mass effect.*

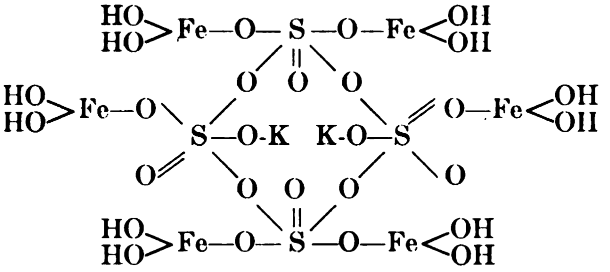
The alunite from Red Mountain, described by Hurlburt, was analyzed in the Sheffield mineralogical laboratory under the direction of one of the present writers, and it was found that water was first expelled from the compound at a rather high temperature, thus indicating that the mineral contains hydroxyl and no water of crystallization; accordingly it was shown that the seemingly complex formula of the mineral, expressed by the ration $Al_2O_3 : K_2O : SO_3 : H_2O = 3 : 1 : 4 : 6$, may be much simplified to $K[Al(OH)_2]_3[SO_4]_2$. In the light of the present investigation it now seems best to abandon the above simple formula and adopt one containing double the number of atoms, in order to make clear the isomorphism between $K_2$, $Na_2$, and Pb. The formulas of the minerals of the group would then be expressed as follows:

| | |
|---|---|
| Alunite | $K_2[Al(OH)_2]_6[SO_4]_4$ or $K_2Al_6[OH]_{12}[SO_4]_4$ |
| Natroalunite | $Na_2[Al(OH)_2]_6[SO_4]_4$ or $Na_2Al_6[OH]_{12}[SO_4]_4$ |
| Jarosite | $K_2[Fe(OH)_2]_6[SO_4]_4$ or $K_2Fe_6[OH]_{12}[SO_4]_4$ |
| *Natrojarosite* | $Na_2[Fe(OH)_2]_6[SO_4]_4$ or $Na_2Fe_6[OH]_{12}[SO_4]_4$ |
| *Plumbojarosite* | $Pb[Fe(OH)_2]_6[SO_4]_4$ or $PbFe_6[OH]_{12}[SO_4]_4$ |

In the case of the lead compound one atom of lead, and in the others two atoms of either potassium or sodium, are combined in complex molecules containing fifty other atoms; hence that the complex of fifty atoms, to the right of the $K_2$, $Na_2$ and Pb in the foregoing formulas, should control or dominate crystallization by virtue of *mass effect*, and condition an isomorphism between such unlike elements as sodium, potassium, and lead is not so surprising as would at first appear.

Having adopted the double formulas, as given above, there are numerous ways of writing developed formulas, of which the following are perhaps the simplest and most satisfactory:





It is interesting to note that although $K_2$, $Na_2$ and Pb play so small a rôle in the alunite-jarosite molecules, the substitution of $Na_2$ for $K_2$ is attended by a marked variation in the angles of the crystals, greater, in fact, than is generally observed in isomorphous replacements. That alunite and jarosite containing potash would be nearly alike in their angles is expected, since crystals of corundum and hematite are surprisingly alike, as shown by the following comparison:

| | Axial length. | $r \wedge r'$ | $c \wedge r$ |
|---|---|---|---|
| | | ° ' | ° ' |
| Corundum, $Al_2O_3$ | 1.3630 | 93 56 | 57 34 |
| Hematite, $Fe_2O_3$ | 1.3656 | 94 00 | 57 37 |

The relations of the minerals of the alunite-jarosite group are as follows:

| | Axial length. | $r \wedge r'$ | | $c \wedge r$ | | Birefringence. |
|---|---|---|---|---|---|---|
| | | ° | ′ | ° | ′ | |
| Alunite | 1.252 | 90 | 50 | 55 | $19\frac{2}{3}$ | Positive. |
| Jarosite | 1.245 | 90 | 45 | 55 | 16 | Negative. |
| Natrojarosite | 1.104 | 85 | 54 | 51 | 53 | Do. |
| Plumbojarosite | 1.216 | 89 | 42 | 54 | 32 | Do. |

From the foregoing table it is seen that the substitution of sodium for potassium in jarosite has brought about greater variation in the angles of the crystals than the substitution of the bivalent metal lead for potassium.

The three minerals, natrojarosite, plumbojarosite, and the Na-K-alunite from Red Mountain, are very interesting when studied together as microscopic mounts, the crystals being practically alike in size and development, and illustrating very beautifully on the one hand the isomorphism of aluminum and iron, on the other the isomorphism of potassium, sodium, and lead. The three substances must have formed under like conditions, and it is believed that they are solfataric products, formed under the combined action of heat and pressure. Being difficultly soluble, they have formed, like many precipitates, as fine crystalline powders.

The three products just mentioned, when heated in closed tubes, behave alike; they suffer no change on gentle heating, but when the temperature is sufficiently high to decompose the chemical molecules, the crystals break up into fine powder or dust, which is carried along by the escaping vapors and deposited for a considerable distance along the sides of the tubes. In addition to water, $SO_2$ and $SO_3$ are copiously given off during decomposition. In the case of natrojarosite, and the same would doubtless hold true for the Na-K-alunite, it is found that after ignition one-fourth of the sulphate radicle has been retained by the alkali metal, and may be extracted by water. In the case of plumbojarosite, however, all of the sulphate radicle is expelled by ignition, doubtless because the ferric-oxide present serves to decompose any lead sulphate which might have a tendency to form. Anglesite, $PbSO_4$, when heated alone in a closed tube suffers no decomposition, but when finely triturated with limonite and heated, acid water is given off. Finely powdered natrojarosite and plumbojarosite are slowly but completely soluble in boiling hydrochloric acid. Plumbojarosite when fused with sodium carbonate on charcoal yields globules of lead and a coating of lead oxide.

It has seemed to the writers best to designate the new compounds described in this article as natrojarosite and plumbojarosite, the names signifying their relation to a well-known species. Other members of this group will doubtless be found, and the name "natroalunite" might be employed to designate the two varieties of alunite from Colorado mentioned on page 37, where the proportion of the soda to the potash molecule is 7 : 4. It is highly probable that a series of alunite-jarosite compounds could be made artificially.

It is with pleasure that the writers acknowledge their indebtedness to Messrs. Turner and Porter for calling attention to the interesting compounds described in this article.

# MINERALS FROM THE CLIFTON-MORENCI DISTRICT, ARIZONA.

By W. LINDGREN and W. F. HILLEBRAND.

In 1902 an examination was made of the Clifton-Morenci copper district in Arizona. Study of the collections proved the presence of several interesting minerals, a brief account of which is here given. The copper deposits at Clifton and Morenci consist partly of irregular or tabular bodies of oxidized ores in Paleozoic limestones, partly of chalcocite ores connected with fissure veins in a granite-porphyry or in the same limestones.

## CORONADITE.

On the dump of a small shaft on the west end of the Coronado vein, three-fourths of a mile west of Horseshoe shaft, fairly large amounts of a dark metallic mineral were found intimately intergrown with quartz and decomposing into limonite. The vein at this end shows no copper minerals, but is said to contain some gold, and its surface ores are reported to have been worked in an arrastre in the early days of the camp. In color this mineral is black and its structure is delicately fibrous. The hardness is about 4 and the streak black with brownish tinge.

A thin section proves it to be opaque, and in reflected light its fibrous and homogeneous structure is well brought out. It cements angular quartz grains and its secondary nature is clearly indicated. In general aspect it is not unlike psilomelane. A preliminary examination showed that it contained the oxides of lead and manganese. As it did not seem to correspond to any known mineral species, an analysis was made, after a partial mechanical separation. The results were as follows:

Long-continued efforts to procure pure material for analysis by the use of heavy solutions were not attended with success. The ultimate product of specific gravity, 5.246 at 22°, yielded on decomposition by hydrochloric acid a residue of from 6 to 7 per cent, which consisted mainly of silica, with a small amount of alumina, etc. Its presence would not have mattered much had it been quite indifferent to acids, but its partial solubility, as shown by the varying amounts undissolved on different trials, and similar varying amounts of alumina and perhaps other minor ingredients found in solution, renders the calculation of molecular ratios not altogether certain in all cases. The composition as found is:

| | |
|---|---|
| $MnO_2$ | [a] 56.13 |
| MnO | 6.56 |
| PbO | 26.48 |
| ZnO | .10 |
| CuO | .05 |
| $MoO_3$ | .34 |
| $Al_2O_3$ | [b] .63 |
| $Fe_2O_3$ [c] | 1.01 |
| $H_2O$ | [d] 1.03 |
| Insol. and silica | 7.22 |
| CaO, MgO, Alk., and loss | .45 |
| Total | 100.00 |

The material available did not permit the quantitative determination of the vanadium, which may be present in rather more than a mere trace, but neither it nor the phosphorus can influence materially the ratios given below. The vanadium would be effective in two ways: (1) by requiring a base for its neutralization, if existing as an acid constituent, and (2) by liberating chlorine when acted on by hydrochloric acid, and thus affecting the values found for peroxide oxygen. If the iron exists in the ferrous state, it too would affect the values found for the peroxide oxygen, and consequently for both the oxides of manganese. Assuming it to so exist and applying the proper corrections, also deducting from the lead oxide an equivalent for the molybdenum, assuming its existence as molybdate of lead, the following are the results:

| | | |
|---|---|---|
| $MnO_2$ | 56.68 ÷ 87.0 = 0.6515 | = 3.00 |
| MnO | 6.11 ÷ 71.0 = .0861 | 0.217 = 1.00 |
| PbO | 26.96 ÷ 222.9 = .1165 | |
| FeO | .91 ÷ 72.0 = .0126 | |
| ZnO | .10 ÷ 81.0 = .0012 | |
| CuO | .05 ÷ 79.0 = .0006 | |
| $H_2O$ | 1.03 ÷ 18.0 = .0572 | = .264 |

---

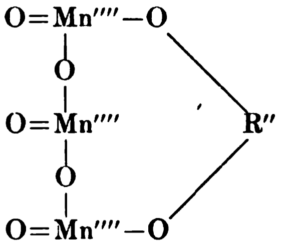
[a] Mean of 56.10 and 56.16. Total Mn as MnO from $MnSO_4$, 52.38 per cent. Peroxide oxygen, 10.31 per cent.

[b] With a little $TiO_2$, $P_2O_5$, and $V_2O_5$.

[c] State of oxidation not known.

[d] Nothing at 100°, only 0.14 per cent below 200°.

If the mineral is to be regarded as anhydrous, the comparatively simple formula $R''(Mn_3O_7)''$ satisfies the above ratio, and it may be written structurally·



in which $R''=Pb''$ or $Mn''$. This is to be regarded as a saturated salt of one of the numerous possible derivatives of ortho-manganous acid that may be derived from it by removal of water, in the present case as follows:

$$3H_4MnO_4-5H_2O=H_2Mn_3O_7$$

An acid of the same empirical formula would result by removal of two molecules of water from three of metamanganous acid, $H_2MnO_3$. It is probably best to rest for the present content with the above relatively simple formula and to regard the water found as due to incipient alteration. But if the water is to be considered as wholly or in part essential, and furthermore constitutional—and this may very well be the proper view to take—then the formula becomes much more complex, namely, $R_4''H_2(Mn_{12}O_{29})$, when none of the water is allotted to the foreign matter. This formula is still referable graphically to a more highly condensed manganous acid, and a number of isomers would be possible.

Such intricate formulas as this should not cause the least surprise, however unlikely they may at first appear to be. The great number of manganites in varying degrees of saturation and hydration observed in nature and prepared artificially, some of them of even greater complexity than the above, are certainly not all mixtures of only a few simply constituted molecules. A very short study of the graphic formula corresponding to the above empirical formula $R''_4H_2(Mn_{12}O_{29})$ will show what a vast number of closely related bodies are theoretically producible by hydrating the molecule step by step or by adding to or reducing the number of bivalent atoms or substituting for them those of another valence. Similar varieties in great number would be derivable from other condensed manganous acids of both higher and lower orders, and it is plain that because of the very slight differences in percentage composition between many of them it is almost as hopeless to expect analysis to reveal the exact empirical formula in the majority of cases as it is for the enormously complex albuminous bodies of organic chemistry. This is especially true because in so many cases the mineral manganites described are far from being homogeneous species. They are either mixtures of two or more of these closely related complex molecules or else are contaminated by foreign bodies. It is not surprising, then, that so many compounds of uncertain formula that may be regarded as salts of manganous acid have been prepared in the laboratory or are found in nature. From the known tendency of these bodies to form under laboratory conditions which may very well be repeated in their general character in nature, it is to be expected that a vast number of mineral manganites should exist, and it ought rather to excite surprise than otherwise if two or more are not formed simultaneously from the same solution. This, together with inherent difficulties of analysis, would offer a simple explanation of the fact that so few of the analyses made lead to rational formulas. If formed from solution, their original state might well be one of hydration either as regards water of crystallization or of constitution. The temperature at which the water is expelled in the present case indicates constitutional water.

Our search of the literature has not revealed a native manganite carrying a high percentage of lead, although artificial compounds have been prepared. For this reason, and because of its distinctly crystalline character, the present mineral seems worthy of a specific name. The one we propose is *coronadite*, after the famous explorer of that portion of the American continent from which the Territories of New Mexico and Arizona have been formed.

## CHALCOCITE.

The cuprous sulphide ($Cu_2S$) is very common in the Clifton district; in fact, it constitutes at present the principal valuable mineral in the ores. It occurs chiefly intergrown with pyrite, in the altered porphyry, as disseminated grains or as solid seams or veins which rarely exceed 2 or 3 feet in thickness. It is never crystallized but has ordinarily an earthy or sooty appearance and black color; scratching with a knife reveals its semisectile character and metallic luster. In a few small massive veinlets the normal metallic luster and dark-gray color appear on fractures; a fibrous or columnar structure of the mineral is known on small seams in shale from the Montezuma mine. The mineral prefers porphyry, and the great bodies of ore now worked all occur in this rock; but it is not entirely unknown from the irregular deposits in limestone generally carrying cuprite and copper carbonates. A partial analysis of massive chalcocite from the Montezuma mine, Morenci, gave 96 per cent $Cu_2S$ and 2.4 per cent $FeS_2$, the latter probably mechanically admixed.

The chalcocite is everywhere, in this district, a secondary mineral formed by the replacement of pyrite by means of descending solutions of cupric sulphate. The deposition of the mineral was accompanied by the formation of quartz, chalcedony, and kaolin. In the porphyry the chalcocite ore along the veins begins 100 to 200 feet below the surface and continues to a depth of 400 feet, or even more, when it is usually replaced by pyrite, chalcopyrite, and zinc blende.

## WILLEMITE.

This rare silicate of zinc ($Zn_2SiO_4$) was found by Mr. Boutwell in the form of very small grayish crystals on a fragment of garnet rock in the Modoc open cut, on the north side of Modoc Mountain. These crystals were identified by Messrs. Pirsson and Penfield, of Yale University, who state that the stout hexagonal prisms look exactly like those from the original locality at Moresnet.

## CALAMINE.

Small transparent orthorhombic crystals of calamine $(ZnOH)_2 SiO_3$ were identified on a specimen of decomposed garnet rock from the Shannon mine, just above the lime quarry.

## DIOPTASE.

The silicate of copper, dioptase ($H_2CuSiO_4$) has been found at only a few localities. Very beautiful specimens, which, however, are by no means common, have long been known from the classic locality, the Kirghese Steppes, Russia, and more recently from the French Kongo State, Africa. Dioptase is seldom found in the United States, the only recorded occurrences being at the Bon Ton mines, Chase Creek, near Clifton, Ariz., noted by R. C. Hills,[a] and from near Riverside, Pinal County, Ariz., noted by W. B. Smith.[b] Well crystallized specimens of this mineral were found on an old dump of the Stevens group of mines, on the west side of Chase Creek, near Garfield Gulch. They occurred in a small chimney of chrysocolla ore in limestone, now worked out, and the locality is believed to be the same as that described by Mr. Hills. The dioptase crystals were submitted to Prof. S. L. Penfield, who remarks on them as follows:

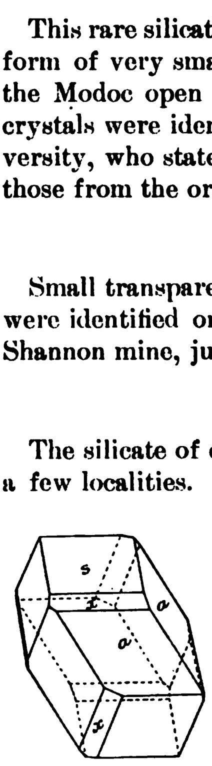


FIG. 3.—Dioptase.

> The crystals, measuring from 1 to 2 mm. in diameter, occur closely grouped together, lining cavities in a brown ferruginous gangue impregnated with amorphous green material which is probably chrysocolla. The color of the dioptase is a beautiful emerald-green. The habit of the crystals, shown by the accompanying figure, is that which is most commonly observed and is especially characteristic for dioptase; prism of the second order *a* $(11\bar{2}0)$, terminated chiefly by the rhombohedron of the first order *s* $(02\bar{2}1)$ and with small faces of the rhombohedron of the third order *x*

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[a] *Am. Jour. Sci., ser. 3, vol. 23, 1882, p. 325.*

[b] Proc. Colorado Sci. Soc., vol. 2, 1887, p. 159.

($13\bar{4}1$). As is common on this species, the prismatic faces are vicinal and the *s* and *x* faces are striated parallel to their mutual intersection edges; hence the crystals are not suited for giving accurate measurements of the angles with the reflection goniometer. One crystal was measured, and the angles of one of the rhombohedral zones, given below, are sufficiently close to the calculated values to establish the identity of the forms.

| | | | Measured. | Calculated. |
|---|---|---|---|---|
| *a x*, | $11\bar{2}0$ | $13\bar{4}1$ | = 28° 55′ | 28° 48′ |
| *s s′*, | $02\bar{2}1$ | $\bar{2}021$ | = 83° 48′ | 84° 33′ |
| *s′ a″*, | $\bar{2}021$ | $\bar{1}\bar{1}20$ | = 48° 18′ | 47° 43′ |

By crushing some of the material, embedded in oil under a cover glass, and examination in convergent polarized light, occasional fragments were found which gave a normal uniaxial interference figure, with numerous rings indicating high birefringence. The character of the birefringence was found to be positive. Thus in all of its crystallographic and optical relations the material studied is like typical dioptase from other localities.

## CHRYSOCOLLA.

This mineral ($CuSiO_3+nH_2O$) occurs very commonly in the oxidized part of the deposits, but does not, except in some cases, constitute an important ore. On the whole, it is more abundant in the deposits in porphyry and granite than in those contained in limestone. The usual bluish green or dirty green colors and conchoidal fracture characterize it. It occurs in seams or coatings at many of the mines—abundantly in the Mammoth mine on contact fissure between porphyry and limestone; at several prospects on the Stevens group in Chase Creek near Garfield Gulch; in the Terazas fissure vein in porphyry, near Metcalf; at the Metcalf mines and many of the prospects between that place and Morenci; at the Modoc open cut, Morenci. Technical analyses of chrysocolla ore from Terazas mine by the Arizona Copper Company gave—

| | |
|---|---|
| $SiO_2$ | 31.65 |
| $CuO$ | 34.90 |
| $H_2O$ | 26.30 |
| $Al_2O_3$ | 3.80 |
| Undetermined | 3.35 |
| | 100.00 |

Normal chrysocolla should have 34.2 per cent $SiO_2$, 45.2 per cent $CuO$, and 20.5 per cent $H_2O$, but the analyses show great divergency, many probably being mixtures. Moreover, what has been called chrysocolla probably includes two mineral species.

The optical characteristics of chrysocolla seem imperfectly known. Dana states that it is cryptocrystalline, while many other text-books, notably one issued in 1902 by Professor Miers, call it "amorphous." In most cases the mineral indeed seems cryptocrystalline with bluish-gray colors of interference, but this is by no means universal.

Chrysocolla from the Modoc open cut appears as mammillary crusts of bluish-green color on "copper-pitch ore." The latter is isotropic and undoubtedly a distinct mineral from the chrysocolla, of brown color in varying tints, some of it opaque and showing evidence of concentric deposition. On top of the chrysocolla are thin crusts of quartz and some calcite. The chrysocolla has three different structural forms, as seen under the miscroscope: (1) The dominant mass is a cryptocrystalline to microcrystalline aggregate of particles with high birefracting index; (2) very fibrous and felted aggregates of same substance giving undulatory effects between crossed nicols and medium high colors; (3) fibrous crusts on top of 1, or also in thin layers between masses of 1, the individuals having such a remarkably parallel orientation that the aggregate of them appears almost like single crystals between crossed nicols, with black shadows sweeping across them when the table is turned. The extinction is parallel to the fibers, double refraction strong, about like augite, character negative. The same optical characteristics were repeatedly observed in thin sections of chrysocolla from Metcalf and other places. Reniform deposits were sometimes noted, the center of cryptocrystalline material coated with coarsely fibrous and highly birefringent material.

Sections from the Coronado and Metcalf mines often showed pseudomorphs of pyrite consisting of a shell of limonite with kernel of fibrous chrysocolla.

The observations of Jannettaz[a] on chrysocolla from Boleo, Baja California, Mexico, led to the same results as described above, but seem generally to have been overlooked by editors of text-books.

## COPPER-PITCH ORE.

Under this old German name is described a dark brown to black substance, sometimes dull but generally with glassy to resinous luster; hardness about 4; streak dark brown. It occurs among the products of oxidation of the deposits in limestone, as at the Detroit and Longfellow mines and Modoc open cut at Morenci, and is associated with azurite, malachite, and chrysocolla, often inclosing these minerals or replacing in branching veinlets, together with azurite, a shale-like mass, probably largely composed of kaolin. In thin section it is sometimes opaque, but often also translucent, gradual transitions obtaining in the same section, and occurs in irregular or concretionary masses, often containing small embedded crystals of a doubtful mineral, possibly a silicate of zinc. Between crossed nicols the translucent mineral always proves entirely isotropic and, except for varying depth of color and the small crystals mentioned, entirely homogeneous.

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[a] Bull. Soc. Min., Paris, vol. 9, 1886, p. 211.

A rough preliminary analysis of selected pitch black material from the Detroit mine gave

| | |
|---|---|
| $CuO$ | 28.6 |
| $ZnO$ | 8.4 |
| $MnO_2$ | 21.2 |
| $Fe_2O_3+Al_2O_3+P_2O_5$ | 4.0 |
| Insoluble in HCl | 22.8 |
| Ignition loss 16.3, less oxygen due to conversion of $MnO_2$ to $Mn_3O_4$. | 13.7 |
| | 98.7 |

Similar material surmounted by crusts of chrysocolla from the Modoc open cut contained much $MnO_2$, with a good deal of CuO and ZnO, and is thus evidently the same substance. Manganese is largely but not certainly wholly present as $MnO_2$. The insoluble portion consists of silica, is wholly separated by acid without need of evaporation, and is nearly all soluble in dilute potassium hydroxide. It is not possible to say whether silica is in combination or as opal, but it can not be present in any other form.

Most of these copper-pitch ores, known from many districts, have been described as impure chrysocolla. As shown by the optical characteristics, they are not, however, a mixture, and they certainly do not contain any chrysocolla, the characteristics of which are very different. They probably represent a series of closely related compounds, the chemistry of which has not yet been fully elucidated. Prof. G. A. Koenig[a] describes a similar mineral with the same isotropic character from Bisbee, and names it melanochalcite. Its composition is different, since it contains

| | |
|---|---|
| $CuO$ | 76.88 |
| $SiO_2$ | 7.80 |
| $CO_2$ | 7.17 |
| $H_2O$ | 7.71 |
| $ZnO$ | .41 |
| $FeS_2$ | .07 |
| | 100.04 |

Professor Koenig considers it most probably a basic salt of an ortho-silico-carbonic acid. No carbon dioxide was found in the Morenci minerals. In conclusion it would seem that the chemistry of these copper-pitch ores would bear further examination.

## MORENCITE.

In a lime shale on the intermediate level of the Arizona Central mine, Morenci, 200 feet below the surface, brownish or greenish spreading mases were found, containing brownish yellow, silky fibrous seams. The inclosing material consists largely of the same material as the seams, but impure and mixed with a little chlorite and pyrite. The whole bears evidence of being a product of oxidation of some contact metamorphic mineral.

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[a] Am. Jour. Sci., ser. 4, vol. 14, Dec., 1902, p. 404.

The fibrous mineral on the seams forms a felted aggregate as seen under the microscope, but it is well individualized and contains few impurities except a little pyrite and chlorite. The minute fibers are brownish yellow and slightly pleochroic, being darker when parallel to the principal section (opposite the behavior of biotite); the birefringence is strong and extinction strictly parallel to the fibers. No mineral corresponding to this has been described, but, although its individual character is beyond doubt, the analysis does not lead to a satisfactory formula. The material for the analysis was picked out carefully under the lens and examined under the microscope it proved satisfactorily pure.

The analysis afforded the results of the first column of figures below. In deducing the molecular ratios of the second column there has been deducted sufficient lime to form apatite with the phosphoric oxide.

| | | | | Molecular ratios. |
|---|---|---|---|---|
| $SiO_2$ | 45.74 | 757 | | = 10.71 or 11 |
| $TiO_2$ | trace | | | |
| $Al_2O_3$ | 1.98 | 019 | 205 | = 2.90 or 3 |
| $Fe_2O_3$ | 29.68 | 186 | | |
| FeO | .83 | 011 | 141 | = 2.00 or 2 |
| MnO | trace | | | |
| CaO | 1.61 | 027 | | |
| MgO | 3.99 | 100 | | |
| $K_2O$ | .20 | 002 | | |
| $Na_2O$ | .10 | 001 | | |
| $H_2O$ 105° | 8.84 | 491 | | = 6.96 or 7 |
| $H_2O$ 150° | .12 | 282 | | = 3.99 or 4 |
| $H_2O$ below redness | 4.27 | | | |
| $H_2O$ redness | .69 | | | |
| CuO | little | | | |
| $FeS_2$ | .66 | | | |
| $P_2O_5$ | .18 | | | |
| | 98.89 | | | |

It would seem from the temperatures at which the water is driven off that this must exist in two conditions, and that four-elevenths of it must be held more securely than the remaining seven-elevenths. The attempt to account for four molecules of water as constitutional, however, led to no simple or seemingly probable formula, whereas if all water is excluded the ratio is that of a metasilicate—$R''_2R'''_6(SiO_3)''_{11}$. On the other hand, to include the whole of the water as essential to the silicate molecule, for which there is little ground in view of the ease with which most of it is expelled, leads to an orthosilicate ratio—$R'_{22}R''_{2}R'''_{6}(SiO_4)''''_{11}$.

On the whole, considering also the bad summation of the analysis, it is more rational to regard the mineral as a hydration product of an original metasilicate molecule than to attempt to construct a complex formula which could have but a very doubtful value. As the mineral is not a mixture, but is optically well individualized, we have, after some hesitation, thought best to designate it by the name *morencite*, derived from the locality in which it was found.

## LIBETHENITE.

This hydrous, basic phosphate of copper ($H_2Cu_4P_2O_{10}$) was found 30 feet below the adit level of the Coronado lode, in the main shoot. It is a matter of interest to record its occurrence, for this rare mineral has never before been noted in the United States. It occurs in small crystals, less than 1 mm. in length, deposited in cavities and seams in a quartzite gangue. The mineral was identified by Prof. S. L. Penfield, who also kindly measured and figured the crystals. Professor Penfield describes the occurrence as follows:

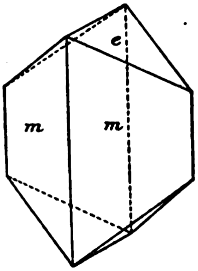

FIG. 4.—Libethenite.

> The only associated minerals are occasional clusters of minute quartz crystals and small tufts of radiated malachite needles. The color of the libethenite varies from light to dark olive-green, depending upon the size of the crystals. The habit of the crystals, as shown by the accompanying illustration, is a combination of the prism *m* (110) and brachydome *e* (011), which is exactly like that commonly observed on libethenite from foreign localities. On an occasional crystal the brachypinacoid *b* (010) was also observed. Although the crystals are brilliant, the faces are generally vicinal and give uncertain or multiple reflections of the goniometer signal. The best reflections were obtained from the faces of the dome *e*, and three measurements of *e* *e′*, 011 $0\bar{1}1$ gave 69° 52′, 70° 18′, and 70° 14′. The last measurement, obtained from the best reflections, is close to the value, 70° 8′, obtained by Rose. The best measurements of the prismatic angle gave $m \wedge m'''$, $110 \wedge 1\bar{1}0 = 87° 11'$, which, considering the vicinal character of the prismatic faces, is reasonably close to the value of Rose, 87° 40′, as given in Dana's Mineralogy. A small crystal resting on a prismatic face, when examined in convergent polarized light, showed an optical axis nearly in the center of the field, with the dark bar running at right angles to the vertical axis, thus indicating that the optical axes are in the plane of the base, as determined by Des Cloizeaux. The presence of copper, water, and phosphoric anhydride was determined by chemical tests.

A more detailed search would probably reveal small quantities of phosphates from other mines near Morenci. They are certainly not abundant.

## BROCHANTITE.

This basic sulphate of copper ($H_6Cu_4SO_{10}$) is usually supposed to be one of the rarer minerals. It was, however, discovered at a few places near Metcalf and Morenci, in well-developed crystals, and this led to a systematic microscopic examination of the green ores, hitherto supposed to be malachite. The result was surprising, as the mineral was proved to be of extremely common occurrence, mostly intergrown with malachite, which had effectively masked its presence. It is believed that a careful examination of many so-called malachites from other districts will disclose the overlooked importance of brochantite as a copper ore.

Brochantite is frequently crystallized in the short but stout rhombic prisms combined with dome and brachypinacoid characteristic of the species. Needle-shaped and flat crystals are more rare. The crystals are usually of small size and frequently microscopic. It occurs as lighter or darker emerald-green crusts on limonite or sericitized porphyry from the red ore body in the Shannon mine, from the Metcalf mines and many other places; as fine-grained aggregates in altered porphyry at the Shannon mine, near the surface, constituting valuable ore with as much as 30 per cent copper; from croppings of the King vein, filling seams and coating porphyry fragments as flat pieces or even foils with almost pearly luster; from the croppings of the Copper Queen mine between Morenci and Metcalf, here as flat stellar aggregates of bluish green foils; at many places near Morenci, as, for instance, Copper Mountain and Montezuma mines, at the latter locality replacing chalcocite. It would probably not be found absent from any mine in the district containing oxidized copper ores. Malachite often develops later than the brochantite.

On the whole, the mineral is most abundant in fissure veins in porphyry, though also occurring in the irregular deposits in limestone.

Brochantite has an excellent cleavage parallel to the brachypinacoid. The macropinacoid is the axial plane and the acute bisectrix is seen emerging in cleavage foils. Pleochroism very slight. Birefringence much lower than malachite, about equal to that of augite. This, as well as the absence of twins, distinguishes brochantite from malachite. The reaction for sulphuric acid is of course a valuable aid.

## SPANGOLITE.

This peculiar mineral ($H_{18}Cu_6AlClSO_{19}$), essentially a highly basic chloro-sulphate of copper and aluminum, was discovered and described by Prof. S. L. Penfield[a] about fifteen years ago. The specimen came from some point within 200 miles of Tombstone, Ariz., and probably from one of the great copper camps of the Territory. Somewhat later it was identified by Prof. H. A. Miers on two specimens from Cornwall, England, but the American locality has not yet been found. It is, therefore, a matter of interest to record its discovery on some specimens from the Metcalf mine of the Arizona Copper Company, taken from the workings in the great open cut not more than 100 feet below the surface. These specimens consist of white sericitized granite-porphyry, in part silicified, and traversed by veinlets and irregular masses of cuprite; the cuprite contains native copper and is covered by crusts of malachite, brochantite, and chrysocolla. A soft and scaly bluish-green coating on the chrysocolla proved to consist of microscopical hexagonal crystals or cleavage foils, remaining dark between crossed nicols. The mineral was identified as spangolite, a determination in which Professor Penfield concurred. No measurable crystals were found, and the mineral is very inconspicuous. It is difficult, if not impossible, to obtain material entirely free from other minerals.

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[a] Am. Jour. Sci., ser. 3, 1890, vol. 39, pp. 370-378.

Selected bluish flakes from this specimen gave tests for water, and the sulphate and chlorine ions, besides copper. There was too little of this pure material to permit of a test for alumina, but the mixed copper minerals composing the greater part of the specimen showed the presence of this body. It seems therefore probable on these grounds alone that the bluish flakes are spangolite. Vanadium, phosphorus, and arsenic are absent.

The closed-tube reactions of the mixed copper minerals are very striking. Water is given off first. Then appears suddenly a white sublimate ($AlCl_3$?) near the assay, which seems to form or at once change to minute colorless drops. This deposit can be driven slowly up the tube, followed at its lower, sharply defined edge by dark yellow-brown drops ($CuCl_2$?), which on cooling solidify to greenish crystalline aggregates, and the part of the tube between them and the assay shows under the lense delicate feathery crystallizations like frost markings on window panes. Down in the flame the glass becomes colored red ($Cu_2O$?) and in parts yellow. On charcoal the blowpipe flame is colored azure blue and at the same time green.

In order to compare the above closed-tube behavior with that of undoubted spangolite, a small fragment of the latter, offered by Professor Penfield, was tested. It gave water and then a white sublimate like the one above mentioned, followed by a dark olive-brown liquid, which on cooling passed through lighter shades of color and solidified as a bright green ring. In general this behavior is very like that of the mixture under examination from Clifton.

## GERHARDTITE.

The cliffs of granite-porphyry in the deeply eroded Chase Creek Canyon at Metcalf in many places show a conspicuous and extensive *bright green coating* of some copper mineral, which, no doubt, is formed by the trickling of atmospheric waters over and through rocks containing a small percentage of copper. This is not surprising, for porphyry in this vicinity is altered throughout by quartz cementation and disseminated cupriferous pyrite. This "green paint," as it is frequently called, is not soluble in water, and more closely examined consists of small dark-green, roughly mammillary forms, coating the rock to a thickness of a few millimeters. Examination by the microscope fails to reveal any recognizable mineral in the cryptocrystalline mass.

Chemical examination led to the interesting result that the copper minerals present consist of a nitrate and a chloride, neither of which has been found elsewhere in the mines of the district. Detrital grains and some silica seem associated with these compounds. The nitrogen seems difficult to account for in the absence or scarcity of animal substances which might have yielded it. Possibly it is contained in the porphyry.

The closed-tube reactions of the copper minerals forming the mixture on this specimen are as striking in their way as those of the mixture containing spangolite, described elsewhere. Water first appears, then brownish nitrous vapors, followed by a sublimate which is not very volatile, becomes black on further heating but on cooling yellow-brown. The glass at the bottom of the tube is often yellow-brown when cold. After some hours the sublimate nearly disappears or becomes greenish from absorption of water. If the water which condenses in the upper part of the tube on first applying heat is driven out by the flame, and the mouth of the tube is held in the flame, this is colored deep green by a volatile copper compound (chloride?). On charcoal the flame is azure blue and at the same time green. Vanadium is absent.

The mixture contains presumably the basic nitrate gerhardtite ($H_6Cu_4N_2O_{12}$) and a chloride which is perhaps atacamite. Spangolite, the chloride, can hardly be present, for the slight amount of $SO_3$ shown by test does not seem sufficient to account for the large amount of chloride.

The only place from which gerhardtite has previously been identified is at Jerome mines in the central part of Arizona, associated with cuprite and malachite. It was discovered there by Messrs. H. L. Wells and S. L. Penfield.

# TWO TELLURIUM MINERALS FROM COLORADO.

By W. F. HILLEBRAND.

## 1. EMMONSITE (?) FROM A NEW LOCALITY.

Messrs. F. L. Ransome and Waldemar Lindgren, of the Geological Survey, collected, in the W. P. H. mine at Cripple Creek, a green mineral which has been observed in other mines there, and which on examination in the laboratory showed close resemblance to the emmonsite described by the writer nearly twenty years ago.[a] It differed from it, however, in outward appearance by assuming mammillary forms instead of crystalline plates. In its optical properties, so far as they were determinable, there is perhaps no positive disagreement with those reported for emmonsite. Mr. W. T. Schaller reports as follows:

> There are two cleavages, one parallel to *b*(010) and another parallel to a form in the orthozone. Axial plane parallel to *b*(010). $Bx_a$ perpendicular to a cleavage face in the orthozone. The extinction on the clinopinacoid is inclined 25° to 30° to the vertical axis. 2E is approximately 40°. Double refraction medium, and the mineral nonpleochroic.

The gangue in which the specimens were found is granite and schist, close to their contact with a porphyritic breccia, in a vein pocket, at a distance of about 150 feet from the surface. Associated with it was very rich native gold ore and also tellurite, though neither of these was apparent on the few specimens that came to the laboratory.

Like emmonsite, the mineral melts at a low heat to a red-brown liquid, but, unlike it, gives on stronger heating only tellurous oxide with no trace of selenium or selenious oxide. Analysis confirmed the absence of selenium. Its density, too, differs from that of emmonsite, if the determinations in both cases on scanty material are to be depended on. After allowing for gangue, the original emmonsite was judged to have a density of at least 5, while that of the present mineral is but little above 4.53, after allowing for 24.44 per cent of gangue, consisting mainly of quartz, and to which the specific gravity of quartz was assigned.

In its appearance the present mineral would seem to resemble durdenite more than emmonsite, but the marked difference in water content differentiates it sharply from that mineral, durdenite yielding over 10 per cent.

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[a] Proc. Colorado Sci. Soc., vol. 2, p. 20, 1885.

After deducting 22.44 per cent of gangue containing over 90 per cent of silica, three portions of from 0.15 to 0.20 gram net weight each gave the following results:

| | 1. | 2. | 3. | Mean. | Ratios. |
|---|---|---|---|---|---|
| $TeO_2$ | 70.83 | 71.80 | 70.20 | 70.71 | 3.16 |
| $Fe_2O_3$ | 22.67 | 22.81 | 22.79 | 22.76 | 1.00 |
| $H_2O$ at 100° | 4.68 | 4.82 | .21 | .21 | ........ |
| $H_2O$ above 100° | | | ........ | 4.54 | 1.77 |
| $P_2O_5$ | ........ | .34 | ........ | .34 | ........ |
| $Al_2O_3$ | ........ | .58 | .54 | .56 | ........ |
| $SiO_2$, etc.[a] | ........ | ........ | ........ | .88 | ........ |
| | ........ | ........ | ........ | 100.00 | ........ |

[a] Includes alkalies, traces of magnesia and gold, and a small amount of a metal or metals precipitable by hydrogen sulphide whose identity could not be established.

Allowing the alumina to offset the $P_2O_5$, though it may belong to a soluble silicate or to the tellurite, and a small portion of iron be demanded for the $P_2O_5$, the ratios given in the final column result. They are as unsatisfactory as those afforded by the original emmonsite, which were for $Fe_2O_3$ to $TeO_2$, 1:3.65 in the original description and 1:3.75:1.82 for $Fe_2O_3$:$TeO_2$:$H_2O$ if the supplementary determinations in Am. Jour. Sci., vol. 40, p. 81, 1899, are accepted. The presence of tellurite in association with the green mineral suggests a possible explanation of the failure to obtain a simple ratio, though such contamination was not noted in the material analyzed nor on the neighboring gangue. If this explanation is correct, however, the variation from the original emmonsite ratio becomes still more marked. Provisionally, the mineral may be regarded as emmonsite.

The above results are given in some detail, notwithstanding their inconclusiveness, because of the importance of accumulating data regarding the as yet small but interesting group of ferric tellurites, and inciting collectors and mining men to careful search for and preservation of further material for more extended study.

Thus far emmonsite, durdenite, and an unnamed mineral from Cripple Creek, described by Knight in Proc. Colorado Sci. Soc., vol. 5, p. 66, and affording likewise unsatisfactory ratios, comprise the list of natural ferric tellurites, the formula of no one of which can be regarded as established beyond question.

## 2. TETRADYMITE.

Specimens received in the Survey laboratory from Mr. H. B. Trader and said to come from some locality near Whitehorn, Fremont County, proved to be tetradymite. They showed the mineral in masses of considerable size, which, although manifesting the eminent cleavage of tetradymite, were devoid of distinct evidences of crystal form. The specific gravity at 20° C. was 7.816. Analysis afforded the following results:

| | 1. | 2. | 3. | Mean. | Atomic ratio. | |
|---|---|---|---|---|---|---|
| Bi | 52.00 | 52.30 | 52.13 | 52.14 | 0.2498 | =2.0 |
| Te | 46.62 | 46.82 | | 46.62 | .3651 | =2.98 |
| Se | [a].20 | | | .20 | .0025 | |
| S | .13 | .14 | | .14 | .0044 | |
| $Fe_2O_3$ | .22 | | | .22 | | |
| Insoluble | .15 | | | .15 | | |
| | | | | 99.47 | | |

[a] Approximate.

# NOTES ON LAWSONITE.

By W. T. Schaller and W. F. Hillebrand.

Crystals of lawsonite are very simple in their combinations, the common forms being the prism, base, and brachydome. Two habits occur—tabular crystals with {001} and {110} and crystals with the forms {110} and {011}. The brachypinacoid and the brachydome {041} also occur. A large number of crystals from the typical locality in Marin County, Cal., were collected by the writer and carefully examined with a hand lens for any additional forms. Only two new forms were determined.

The forms observed on the six crystals measured are:

$c = 0 = 001$　　$d = 01 = 011$
$b = 0\infty = 010$　　$e = 04 = 041$
$m = \infty = 110$　　$r = 2 = 221$
$s = 3 = 331$

The angles measured, with those calculated for these forms, are quoted in the table following:

| No. | Letter. | Symbol. | | Measured. | | Calculated. | |
|---|---|---|---|---|---|---|---|
| | | Gdt. | Miller. | φ. | ρ. | φ. | ρ. |
| | | | | ° ′ | ° ′ | ° ′ | ° ′ |
| 1 | *c* | 0 | 001 | .. .. | 0 02 | .. .. | 0 00 |
| 2 | *b* | 0∞ | 010 | 0 00 | 90 00 | 0 00 | 90 00 |
| 3 | *m* | ∞ | 110 | 56 22 | 90 00 | 56 22 | 90 00 |
| 4 | *d* | 01 | 011 | 0 09 | 36 36 | 0 00 | 36 27 |
| 5 | *e* | 04 | 041 | 0 00 | 71 18 | 0 00 | 71 18 |
| 6 | *r* | 2 | 221 | 56 27 | 69 10 | 56 22 | 69 27 |
| 7 | *s* | 3 | 331 | 56 22 | 76 21 | 56 22 | 75 58 |

The new pyramid $r=2=\{221\}$ is present on two crystals, the faces giving good reflections. Fig. 1 shows one crystal on which this form occurs.

The new pyramid $s=3=\{331\}$ is present on only one crystal and but one face of the form occurs. The relative size of the face is about the same as that of the preceding pyramid. The reflection was fair.

The brachydomes are usually deeply striated, rendering it difficult to determine with certainty any domes present. On two crystals reflections were obtained from two faces, measurements of which agree with the angles calculated for the form $\{034\}$. The form is rather uncertain and is not included with the others.

| Measured. | | Calculated. | |
|---|---|---|---|
| $\phi$ | $\rho$ | $\phi$ | $\rho$ |
| 0° 23′ | 28° 07′ — 14′ | 0° 00′ | 28° 59′ |
| 0° 00′ | 28° 07′ | | |

The combinations observed on the crystals measured are shown in the following table:

| Cryst. No. | *c* | *b* | *m* | *d* | *e* | *r* | *s* |
|---|---|---|---|---|---|---|---|
| 1 | .. | .. | *m* | *d* | .. | *r* | .. |
| 2 | .. | *b* | *m* | *d* | .. | .. | *s* |
| 3 | .. | .. | *m* | *d* | .. | *r* | .. |
| 4 | *c* | *b* | *m* | *d* | *e* | .. | .. |
| 5 | .. | *b* | *m* | *d* | .. | .. | .. |
| 6 | *c* | .. | *m* | *d* | *e* | .. | .. |

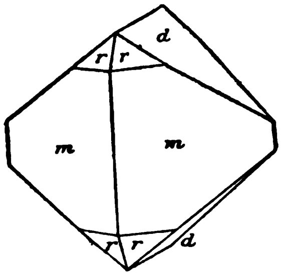

FIG. 5.—Lawsonite.

The following table is a calculation of the two new forms corresponding to the tables given in Goldschmidt's Winkeltabellen.

| No. | Letter. | Symbol. | Miller. | $\phi$. | $\rho$. | $\xi_0$. | $\eta_0$. | $\xi$. | $\eta$. | $x$ (Prism) $(x:y)$. | $y$. | $d=tg\ \rho$. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | ° ′ | ° ′ | ° ′ | ° ′ | ° ′ | ° ′ | | | |
| 6 | *r* | 2 | 221 | 56 22 | 69 27 | 65 45 | 55 54 | 51 13 | 31 14 | 2.2204 | 1.4770 | 2.6667 |
| 7 | *s* | 3 | 331 | 56 22 | 75 58 | 73 17 | 65 42 | 53 53 | 32 30 | 3.3306 | 2.2155 | 4.0001 |

Material carefully purified by the Thoulet solution followed by repeated electromagnetic extraction, and having then a specific gravity of 3.121 at 25°, gave the following results on analysis:[a]

| | | Molecular ratio. | Calculated for the formula $H_4CaAl_2Si_2O_{10}$. |
|---|---|---|---|
| $SiO_2$ | 38.45 | 1.98 | 38.34 |
| $TiO_2$ | .38 | | |
| $Al_2O_3$ | 31.35 | .97 | 32.44 |
| $Fe_2O_3$ | .86 | | |
| FeO | .10 | 1.00 | 17.80 |
| MnO | faint tr. | | |
| CaO | 17.52 | | |
| MgO | .17 | | |
| $K_2O$ | .23 | | |
| $Na_2O$ | .06 | | |
| $H_2O$ (ignition) | 11.21 | 1.95 | 11.42 |
| Total | 100.33 | | 100.00 |

Titanium is not considered in the ratio because probably present as an inclusion of rutile or titanite. If the latter, the ratio would perhaps approximate still more closely to the theoretical than it does. The agreement with the formula deduced by Ransome and Palache from their rather widely differing analyses is very satisfactory. The behavior of the mineral before the blowpipe is somewhat different from that given by those authors, or rather their statement needs amplification. On first applying the flame a splinter appears to fuse easily, as stated, and there is formed a blebby glass, or on larger splinters a porous sinter, but this fusing is only momentary, and it requires the highest heat attainable, under which the fragment emits quite an intense light, to produce a further softening and rounding of the edges. If a rather large splinter is held in the flame of the blowpipe, or in a small flame of a blast lamp, a very sudden and marked exfoliation is observed, but even the extruded points and edges do not fuse completely in the highest attainable heat. The semifused surface, however, appears dark on cooling, sometimes nearly black where the heat was most intense. In producing this exfoliation, if care is taken to apply the flame but for a moment, it has been noticed that a singularly shaped excrescence may shoot out from a point of the surface.

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[a] Concerning this purified material, Doctor Ransome reports as follows:

"I should say the material is as pure as it is possible to get it. The grains are all lawsonite, but vary in individual purity. Some are perfectly clear. Others have minute inclusions, which appear in most cases to be solid particles but are too minute for identification. One of them, however, was suggestive of rutile. Some of the grains are slightly clouded with a yellowish stain which the microscope is unable to resolve into distinct particles. There are apparently a few fluid inclusions also."

# THE COMPOSITION OF YTTRIALITE, WITH A CRITICISM OF THE FORMULA ASSIGNED TO THALÉNITE.

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By W. F. HILLEBRAND.

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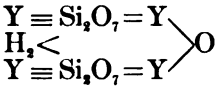
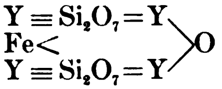
In a paper describing the new yttrium silicate thalénite,[a] the author, C. Benedicks, casts doubt on the formula assigned by Hidden and Mackintosh[b] to their mineral yttrialite from Llano County, Tex., for which they deduced the formula $R_2O_32SiO_2$ or $R_2Si_2O_7$, in which $R_2O_3$ includes the sesquioxide equivalents of very considerable percentages of monoxides and dioxides. Benedicks would assign yttrialite to a group of which rowlandite is the prototype and to which he believes thalénite and kainosite belong, these being either plain basic salts of $H_6Si_2O_7$ of the type $R''R'''_4Si_4O_{15}$ or derivatives in which the fifteenth oxygen atom is replaced by its equivalent in fluorine (rowlandite) or $CO_3$ (kainosite), as shown below.



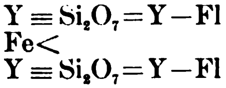
$$\left.\begin{array}{l} Y\equiv Si_2O_7=Y \\ H_2< \\ Y\equiv Si_2O_7=Y \end{array}\right> O$$

Thalénite.



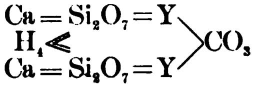
$$\left.\begin{array}{l} Y\equiv Si_2O_7=Y \\ Fe< \\ Y\equiv Si_2O_7=Y \end{array}\right> O$$

Yttrialite.



$$\begin{array}{l} Y\equiv Si_2O_7=Y-Fl \\ Fe< \\ Y\equiv Si_2O_7=Y-Fl \end{array}$$

Rowlandite.



$$\left.\begin{array}{l} Ca=Si_2O_7=Y \\ H_4\leqslant \\ Ca=Si_2O_7=Y \end{array}\right> CO_3$$

Kainosite.

Benedicks says:

> Dem Yttrialit, von Hidden and Mackintosh beschrieben, sollte die Formel $R_2O_3$, $2SiO_2$ zukommen, worin R hauptsächlich Ytter- und Thorerde ist. Dabei wird aber ca. 4% Eisenoxydul in der Analyse vernachlässigt. Wird dies nebst etwas Kalk und Bleioxyd mitgerechnet, so bekommt man die Formel $Fe''O$, $2R_2O_3$, $4SiO_2$, analog mit der des Thalénits, welche besser die Zusammensetzung des Yttrialits wiedergibt, obgleich die Ubereinstimmung gar nicht gut ist.

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[a] Bull. Geol. Inst. Upsala, 4, 1898, 1.

[b] Am. Jour. Sci., ser. 3, vol. 38, 1889, p. 477.

It is, however, not true that Hidden and Mackintosh neglected to take account of the ferrous iron, etc., of their analysis. It was regarded in the derivation of their empirical formula $R_2Si_2O_7$. How wholly unwarranted was the substitution by Benedicks of his formula for yttrialite is shown by the molecular ratio for $RO:R_2O_3:SiO_2$, which he gives as 1:2:4 instead of 1:3.25:7.42, as calculated by me from Mackintosh's figures, wherein for a sound reason I have converted his $UO_3$ into $UO_2$, thus throwing it with the $ThO_2$, where it naturally belongs. The agreement is indeed "gar nicht gut."

Benedicks has made the grave mistake of counting Mackintosh's monoxide bases a second time, thus making a basic salt $R''R'''_4Si_4O_{15}$ instead of the normal one $R'''_4Si_4O_{14}$, to which Mackintosh's results closely conform.

## DISCUSSION OF BENEDICKS'S FORMULA FOR THALÉNITE.

Moreover, in the light of the data furnished by Benedicks himself it can not be admitted that the formula $H_2Y_4Si_4O_{15}$ for thalénite is established.

Water was determined by him according to Penfield's method,[a] but without any hint as to the particular modification employed. If, as seems probable, the water expelled from the mineral was caused to recondense in the cooler part of the ignition tube, the latter being then weighed, and again after driving the condensed water out, two serious sources of error have to be considered: (1) The $CO_2$ present in the mineral, which would count in part as water unless a very careful correction was made, as provided for by Penfield. No mention is made by Benedicks of any such correction. (2) Nitrogen and helium are said to comprise 1.4 per cent of the mineral by weight. If so, these would introduce an error in the above water determination of contrary sign to that due to $CO_2$, and if the proportion of helium were large this error might be of very considerable magnitude.

In an appendix to his paper Benedicks gives an analysis of what he considers to be a very pure form of thalénite. He makes no comparison of this with his earlier analysis, nor does he deduce a molecular ratio, which I find to be 1:2.6:5.15, or 1:3.03:6.02 if small amounts of lime, magnesia, and soda are neglected, instead of 1:2:4, as required by his formula. There being no $CO_2$ in this purer material, the value for water (if determined as above surmised) may be supposed to be affected only by the error due to nitrogen and helium. It will be seen that the neglect to regard lime, magnesia, and soda in his second analysis affects the ratio very seriously. This neglect may be justified in figuring on his first analysis because of an approximate balancing by $CO_2$, but it would be by no means so in the other, in spite of the very satisfactory ratio obtained and leading to the empirical formula

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[a] Am. Jour. Sci., ser. 3, vol. 48, 1894, p. 31.

$R''R'''_6\,Si_6O_{22}$, which is susceptible of a variety of interpretations. It may represent a basic salt of diorthosilicic acid $R''R'''_5(R'''O)'(Si_2O_7)_3$ or of metasilicic acid $R''R'''_2(R'''O)'_4(SiO_3)_6$, or possibly even of other acids.

Finally, if the values for nitrogen and helium are really anywhere near so great as given, an additional argument against the validity of his formula is furnished. For in the light of Kohlschütter's recent researches[a] and my own less conclusive work of a much earlier date,[b] it is in the highest degree probable that nitrogen and helium are not occluded in uraninite and other minerals, but are in chemical combination. Now, if this is so, in a mineral containing as much as 1.4 per cent of nitrogen by weight, this must, quite irrespective of its form of combination, play so important a rôle in the molecule as to utterly invalidate any formula based on calculations from which it is omitted. If the above percentage is made up in large part of helium, its effect, because of its low atomic weight, must be vastly greater than that of nitrogen.

Until light is thrown on the nature of the combinations these two gases form in minerals, no very positive conclusions can be reached as to the formulas to be assigned to those minerals which contain them in more than traces.

## CHEMICAL INVESTIGATION OF YTTRIALITE.

At the earnest request of Mr. W. E. Hidden, the discoverer of yttrialite, I undertook to reanalyze the mineral in order if possible to settle definitely the question of its composition. This seemed especially desirable since a large quantity of very fine material was available.

The appearance and behavior of the mineral agreed in all respects except one with those of the original description.[c] It is there stated that the strongly ignited mineral is insoluble in acid. This is a mistake, for when powdered its solubility in hydrochloric acid is even then perfect, although not rapid.

Careful examination of thin sections under the microscope showed a condition that augured ill for decisive analytical results despite the apparently fine quality of the large specimens. Distinctly foreign mineral fragments were as good as absent except for insignificant coatings of a white alteration product, presumably a carbonate, but considerable shading was apparent in the slides, indicative of alteration or intimate contamination in the mass of the mineral itself. However, after treatment with hot dilute hydrochloric acid (whereby much yttrialite was dissolved) followed by dilute sodium carbonate, the clear glassy residue appeared to be improved in appearance and the specific gravity of two samples, each composed of small grains uniform in size for each sample, had risen from 4.596 and 4.590 to 4.654 at 23½° C. and 4.646 at 26° C., respectively. The two sizes were then mixed, giving one sample of about 4.65 sp. gr. at 25° C. That of the unpurified material analyzed by Mackintosh was 4.575.

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[a] Ann. der Chem., 317, 1901, p. 158.

[b] Bull. U. S. Geol. Survey No. 78, 1891, pp. 76-78.

[c] Am. Jour. Sci., ser. 3, vol. 38, 1889, p. 477.

Qualitative tests showed the absence of zirconium, glucinum, and aluminum and the presence of a little more than a trace of fluorine, of a very little carbon dioxide, besides some other gases which were set free by treatment with acids and by fusion with an alkali carbonate.

Silica was separated by two evaporations with hydrochloric acid; lead was then thrown out by hydrogen sulphide.

The earths when finally collected together free from all iron, manganese, uranium, titanium, phosphorus, calcium, and magnesium, were ignited and weighed. No filtrate from a precipitate of earth oxalates was ever regarded as free from earths till after evaporating, igniting, and retesting. This is a needed precaution in all similar analyses.

The combined earths were dissolved in nitric acid and evaporated to dryness. A saturated solution of potassium sulphate was poured upon the dry mass, which wholly dissolved but almost at once began to deposit double sulphates. Solid potassium sulphate was then added in crystals. After twenty-four hours the precipitate was collected on a filter and washed with the precipitant solution. Twice was this precipitation repeated after first dissolving the double salts in acid, precipitating by potassium hydroxide, washing, redissolving in nitric acid, and evaporating. Then only was the extraction of the yttrium-erbium group practically complete. From the combined filtrates the soluble earths were thrown out, reconverted into nitrates, and again treated with potassium sulphate, whereby a little further insoluble matter was obtained. Yttrium-erbium oxides obtained from the oxalates gave a light-colored mixture of 265.6 Mol. W. (108.8 At. W.), which furnished a pink nitrate solution showing the erbium absorption bands strongly marked, with very faint indications of the strongest band of the didymium components.

The insoluble sulphates were converted into chlorides and thrice treated with potassium hydroxide and chlorine to precipitate thorium and cerium. From the filtrates the soluble earths were recovered and subjected to a repetition of this treatment. Their oxides after purification were light dirty brown when heated over the Bunsen flame, but grayish white when blasted. Their Mol. W. was 335.5 (At. W. 143.7); their nitrate solution was pink and gave the characteristic didymium component absorption bands altogether free from those of the erbium constituents.

Cerium and thorium were separated by a combination of the ammonium oxalate and thiosulphate methods. The weighed thoria was pure white, the ceria of a pale salmon color.

The condition of the uranium was shown by its precipitability as tetrafluoride on dissolving the mineral in hydrofluoric acid. This reaction afforded a ready means of estimating with exactness the ferrous iron by permanganate after rapidly filtering off the fluorides precipitated in an atmosphere of carbon dioxide. Solution of the mineral in a sealed tube with dilute sulphuric acid allowed of finding the oxygen value of both $UO_2$ and FeO by permanganate. The result thus found for $UO_2$ agreed marvelously well with that calculated from the $U_3O_8$ found gravimetrically in a separate portion.

| | Hillebrand. | | Mackintosh. | |
|---|---|---|---|---|
| | | Molecular ratios. | | Molecular ratios. |
| $SiO_2$ | 29.63 | 0.4938 | 29.17 | 0.4861 |
| $TiO_2$ | .05 | | | |
| $ThO_2$ | 10.85 | .0470 | 12.00 | .0483 |
| $UO_2$ | [a]1.64 | | [b].79 | |
| $Ce_2O_3$ | 3.07 | .1930 | 1.86 | .1864 |
| $La_2O_3$, etc | [c]5.18 | | [d]2.94 | |
| $Y_2O_3$, etc | [e]43.45 | | [f]22.67 | |
| " | | | [g]5.30 | |
| " | | | [h]4.50 | |
| " | | | [i]14.03 | |
| $Al_2O_3$ | | | .55 | |
| $Fe_2O_3$ | .76 | | | |
| FeO | 1.96 | .0608 | | |
| MnO | .88 | | 2.89 | .0655 |
| PbO | .80 | | .77 | |
| CaO | .67 | | .85 | |
| MgO | .16 | | .60 | |
| $H_2O$ above 105° C | .32 | | .79 | |
| $H_2O$ below 105° C | .04 | | | |
| $CO_2$ | .11 | | | |
| $P_2O_5$ | .12 | | | |
| N, He, Fl, Alk diff | .31 | | | |
| | 100.00 | | 99.75 | |

[a] Volumetrically. Gravimetrically 1.62 per cent.
[b] Given as .83$UO_3$ by Mackintosh.
[c] At. W. 143.8 (Mol. W. 335.6).
[d] At. W. 162 (Mol. W. 372).
[e] At. W. 108.8 (Mol. W. 265.6).
[f] At. W. 110.3 (Mol. W. 268.6).
[g] At. W. 110.53 (Mol. W. 269.06).
[h] At. W. 114.9 (Mol. W. 277.8).
[i] At. W. 120 (Mol. W. 288).

Nitrogen (?) and helium (?) were obtained quantitatively by fusing the mineral with sodium-potassium carbonate in a current of carbon dioxide and collecting the gases in a nitrometer over potassium hydroxide. The volume was between one and two cubic centimeters per gram of yttrialite. The gases were not further examined.

The analysis is given above, together with that of Mackintosh for comparison.

The ratios of Mackintosh's analysis as calculated above by me are certainly wrong in so far as they are affected by the value for iron, which he assumed to be wholly ferrous. If corrected in accordance with the statement of my analysis, or, what amounts to the same thing for the purpose of illustration and is simpler, if my ratio is altered to conform to his statement for RO and $R_2O_3$ bases, it becomes .0644: .1882—a very close agreement.

It is altogether probable that Mackintosh's separations of the earths were not so far reaching as mine, and this belief is borne out by the differences in the experimental molecular weights for the lanthanum and yttrium groups, mine being more in accordance with what might be expected and, moreover, agreeing almost exactly with those which were found by me for rowlandite in 1893, namely, 336.8 and 266.2 respectively.[a]

It is of course impossible to say what disposition should be made of the small amounts of firmly held water, phosphorus, carbon dioxide, fluorine, and alkalies. The ratios of my analysis are, therefore, to a slight extent incorrect, but probably not enough to influence any conclusions that may be drawn. One thing is apparent, that the preliminary purification by acid has had no pronounced effect on the composition of the mass acted on, otherwise Mackintosh's and my analyses should show far greater differences in the main constituents.

The crude empirical formulas deducible from the ratios of the two analyses are nearly

Hillebrand .................. $R''_{61}$ $R'''_{386}$ $R''''_{47}$ $(Si_2O_7)_{247}$
Mackintosh.................. $R''_{65}$ $R'''_{373}$ $R''''_{48}$ $(Si_2O_7)_{243}$

---

[a] The three minerals gadolinite, yttrialite, and rowlandite occur in Llano County in most intimate association, suggestive of close community of origin, a suggestion which is emphasized by the marvelous agreement for gadolinite and yttrialite, not only in the relative proportions of the trivalent earth metals but in their absolute amounts as well.

| | $Ce_2O_3$ | $La_2O_3$, etc. | $Y_2O_3$, etc. |
|---|---|---|---|
| Gadolinite (Genth) | 2.65 | 5.22 | 44.35 |
| Do | 2.66 | 5.01 | 4.45 |
| Gadolinite (Eakins) | 2.62 | 5.22 | 41.55 |
| Yttrialite (Hillebrand) | 3.07 | 5.18 | 43.45 |
| Rowlandite (Hillebrand) | 5.06 | 9.34 | 47.70 |

*This concordant* testimony of three analysts may be regarded as strong evidence of the correctness *of the earth separations* made by them in these cases. Nearly the same relation is shown by the *trivalent earth-metals* of rowlandite, as seen in the table above.

there being a slight *deficiency* of oxygen atoms in each case for the radical $Si_2O_7$, which is increased by allowing for the $CO_2$ and $P_2O_5$.

In so far then as the character of the acid radical is concerned the results of Mackintosh's analysis are fully confirmed, and there is absolutely no ground for accepting Benedicks's basic formula, which, as I have already shown (p. 62), is based on a palpable error. But the ratios are not at all such as to lend themselves to ready resolution into isomorphous salts of the acid $H_6Si_2O_7$. By doing quite unwarranted violence to the analytical data the above formulas might be reduced to

$$R'' \quad R'''_6 \quad R'''' \quad (Si_2O_7)_4$$

which can be readily represented structurally as a single complex molecule or as a mixture of molecules like $3R'''_2Si_2O_7+R''R''''Si_2O_7$. In neither case, however, is the type of the rowlandite molecule approached, which requires an altogether different ratio of monoxide, dioxide, and trioxide bases; nor, if the second be accepted, is it at all clear that the two molecules would be mineralogically equivalent—that is, isomorphous.

An alternative hypothesis is to regard the mineral as a mixture containing the anhydrous thorite molecule. Proceeding on this assumption and deducting all thorium and uranium and the proper amounts of silicon and oxygen, the crude empirical formulas become

| | |
|---|---|
| Mackintosh | $R''_{65} \quad R'''_{373} \quad Si_{438}O_{1501}$ |
| Hillebrand | $R''_{61} \quad R'''_{386} \quad Si_{447}O_{1533}$ |

which may be interpreted as basic salts of metasilicic acid:

$$R''_{65} \quad R'''_{185} \quad (R'''O)'_{187} \quad (SiO_3)_{438}$$
$$R''_{61} \quad R'''_{194} \quad (R'''O)'_{192} \quad (SiO_3)_{447}$$
$$\text{or} \quad R'' \quad R'''_3 \quad (R'''O)'_3 \quad (SiO_3)_7$$

this last being easily susceptible of symmetrical representation in graphic form.

On the whole, I prefer to leave the constitution of yttrialite unsettled until further evidence can be gathered, either from analyses of allied minerals or from yttrialite itself of more certain purity than any that has yet been discovered.

It must not be forgotten that the gases other than $CO_2$ contained in the mineral may be the cause of the inability to arrive at satisfactory conclusions in the case of this and all other minerals which contain them, as I have already pointed out on page 63.

My excuse for such a lengthy publication on a matter still unsettled is the desire to prevent general acquiescence in the grouping under one type of minerals which can by no means be regarded as proved to belong to that type, and to which yttrialite certainly does not belong.

## SUMMARY.

The empirical formula of Hidden and Mackintosh for yttrialite is confirmed, and it is shown that the basic formula of Benedicks rests on error and has no standing.

The formula of Hidden and Mackintosh is not, however, susceptible of representation as a simple salt of the acid $H_6Si_2O_7$. On the purely hypothetical assumption of admixture of anhydrous thorite, the remaining constituents afford ratios conforming quite closely to those of a basic metasilicate $R'' R'''_3(R'''O)'_3 (SiO_3)_7$.

It is shown for two reasons fully discussed that the formula proposed by Benedicks for thalénite is to be regarded as doubtful.

# A PSEUDO-SERPENTINE FROM STEVENS COUNTY, WASHINGTON.

By F. W. Clarke.

In the course of an investigation upon the ornamental stones of Stevens County, Wash., Mr. R. W. Thatcher and Prof. Elton Fulmer, of the State Agricultural Experiment Station, examined a supposed serpentine which proved to have quite anomalous composition. To the unaided eye the rock was ordinary serpentine of a typical yellowish green color; it was fairly homogeneous, and capable of receiving a fine polish. Incomplete analyses, however, by both of the above-named chemists showed that it was not serpentine, and as I had been consulted with regard to the interpretation of the data, a sample was sent to me for more exhaustive study.

According to Prof. S. Shedd of the Washington Agricultural College, who is conducting the investigation, the rock is from the quarry of the United States Marble Company, 12 miles north and west from Valley, a station on the Spokane Falls and Northern Railroad. It outcrops on the face of a high bluff at an elevation of about 4,070 feet above sea level, and forms a wedge-shaped mass cutting across the mountains in a direction 5° west of north. The adjacent rocks are slates, which lie conformably upon a very coarsely crystalline, dark, almost black, magnesian carbonate. The "serpentine" itself varies a good deal in color, and a series of samples in the United States National Museum show that the output of the locality is far from uniform. They range from a white carbonate, through various intermediate mixtures of the verde antique type, to material which appears to be ordinary serpentine. The latter, however, as shown by the serpentine under consideration, is distinctly laminated in structure, and exhibits a splintery fracture. An analysis by Mr. George Steiger gave the following results:

| | |
|---|---|
| $SiO_2$ | 13.08 |
| $Al_2O_3$ | 1.63 |
| $Fe_2O_3$ | 1.25 |
| $FeO$ | .19 |
| $MgO$ | 56.44 |
| $CaO$ | .33 |
| $H_2O$ at 100° | .85 |
| $H_2O$ above 100° | 23.94 |
| $CO_2$ | 2.03 |
| | 99.74 |

These figures at once suggest a probable admixture of brucite with the serpentinous material, and a microscopic examination by Mr. J. S. Diller tends to confirm this supposition.

According to Mr. Diller the specimen is mainly composed of three minerals, *a*, *b*, and *c*. The first two have nearly equal indices of refraction, but differ widely in birefringence. The mineral *a* is the most abundant. In transmitted light it is colorless, but between crossed nicols it exhibits brilliant colors. The mineral *b* is pale green, and intermingled with *a*. Its birefringence yields weak colors, quite characteristic of chlorite. In quantity it is less than one-fourth of *a*.

Mineral *c* is granular, in scattered grains and irregular groups with a high index of refraction and a birefringence which suggests a carbonate, but it does not effervesce with a dilute acid. It amounts to not more than 5 per cent of the whole. Mineral *b* is certainly chlorite, and *a* may be brucite or possibly serpentine.

By applying Mr. Diller's observations to the analysis of the rock, the proximate composition of the latter may be deduced; although certain assumptions must be made. The carbonate present is probably hydromagnesite, for that species is a frequent associate of brucite. The composition of the chlorite is unknown, but it may be interpreted as essentially clinochlore, and as proportional to the ferric oxide and alumina. So much assumed, the analysis gives the following approximate results expressing the composition of the specimen:

| | |
|---|---|
| Hydromagnesite | 5 |
| Chlorite | 14 |
| Serpentine | 20 |
| Brucite | 60 |
| Extraneous water | 1 |
| Total | 100 |

In this, four minerals appear instead of three, but the microscopic examination did not attempt to discriminate between the brucite and the serpentine.

In order to obtain evidence confirmatory of the foregoing conclusions, a few experiments were made, tending toward fractional determinations. Upon digesting the powdered rock for two hours with cold, dilute nitric acid (10 per cent by volume), 1.32 per cent of $Fe_2O_3+Al_2O_3$ and 47.29 per cent MgO went into solution. In a similar experiment with cold, 20 per cent acetic acid, 0.69 $Fe_2O_3+Al_2O_3$ and 45.64 MgO were extracted. Brucite dissolves readily in acids of the indicated strength, but some chlorite was evidently attacked as well. I also found that ordinary serpentine was quite appreciably acted upon by weak acetic acid. These experiments, then, merely show that the rock contains a large amount of magnesium in a very easily soluble condition, the quantity equivalent to 60 per cent of brucite being 41.4. The results are in harmony with the conclusions already reached, and help to support them, although accurate fractional determinations can not be made. The rock is unusual in character, and if the sample examined is fairly characteristic of the entire deposit, the latter should be carefully studied in reference to its origin and its geological relations.

# ON "CALIFORNITE."

By F. W. Clarke and George Steiger.

In 1901 Mr. G. F. Kunz[a] called attention to a remarkable massive variety of vesuvianite, discovered in California by Mr. A. E. Heighway, and which at first was thought to be jade. A second locality for the mineral, remote from the first, was found some time later, and here the vesuvianite was accompanied by another substance, which proved upon examination to be an unusual modification of garnet. For the vesuvianite Mr. Kunz proposed the varietal name "californite," and at first the garnet was supposed to be the same species, but different in color. Samples of all three minerals were sent to the laboratory of the United States Geological Survey for analysis, and the results obtained are given below.[b]

A. Vesuvianite from the south fork of Indian Creek, 12 miles from Happy Camp, Siskiyou County, Cal.

B. Vesuvianite from Fresno County, Cal., 35 miles east of Selma.

C. White garnet, found with or near vesuvianite B.

| | A. Vesuvianite. | B. Vesuvianite. | C. Garnet. |
|---|---|---|---|
| $SiO_2$ | 35.86 | 36.55 | 38.59 |
| $Al_2O_3$ | 18.35 | 18.89 | 22.24 |
| $Fe_2O_3$ | 1.67 | .74 | .45 |
| FeO | .39 | .74 | .36 |
| MnO | .05 | none | .10 |
| MgO | 5.43 | 2.33 | .64 |
| CaO | 33.51 | 35.97 | 35.97 |
| $H_2O$ at 105° | .29 | .58 | .31 |
| $H_2O$ above 105° | 4.18 | 3.42 | .80 |
| $TiO_2$ | .10 | none | none |
| $CO_2$ | | .91 | .39 |
| $P_2O_5$ | .02 | | |
| F | none | .13 | .17 |
| | 99.85 | 100.26 | 100.02 |
| Less O=F | | .05 | .07 |
| | | 100.21 | 99.95 |
| Specific gravity | | 3.359 | 3.586 |

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[a] Mineral Resources U. S. for 1901, U. S. Geol. Survey; 1902, p. 747. Also, more fully, Am. Jour. Sci., 4th ser., vol. 16, 1903, p. 397.

[b] Analyses by George Steiger.

These analyses may be reduced to uniform type as follows: Throw out hygroscopic water, $P_2O_5$, and $CO_2$, with the equivalent amount of lime representing calcite. Recalculate $TiO_2$ to equivalent $SiO_2$; $Fe_2O_3$ to $Al_2O_3$; FeO, MnO, and MgO to CaO, and F to $H_2O$. Then recompute to 100 per cent. The reduced analyses are as follows:

| | Analyses. | | | Molecular ratios. | | |
|---|---|---|---|---|---|---|
| | A. | B. | C. | A. | B. | C. |
| $SiO_2$ | 35.58 | 37.28 | 39.09 | 0.589 | 0.617 | 0.647 |
| $Al_2O_3$ | 19.24 | 19.74 | 22.82 | .188 | .193 | .223 |
| CaO | 41.03 | 39.42 | 37.22 | .731 | .703 | .664 |
| $H_2O$ | 4.15 | 3.56 | .87 | .231 | .197 | .048 |
| | 100.00 | 100.00 | 100.00 | ........ | ........ | ........ |

Analysis C gives the garnet ratios quite sharply, but as the mineral was white and massive, somewhat resembling chalcedony, its nature was at first unsuspected. Its density is that of garnet, however, its fusibility is the same, and in thin section under the microscope it is optically isotropic. It is undoubtedly garnet containing as an impurity about 1 per cent of calcium carbonate.

Analyses A and B are less easily interpreted. B gives approximately the empirical formula: $2H_2O$, $2Al_2O_3$, 7CaO, $6SiO_2$; or, constitutionally,

$$Al_2Ca_7\ (SiO_4)_6\ (AlOH)\ (AlO_2H_2)\ H.$$

If part of the water is extraneous, the following much simpler formula may hold:

$$Al_2Ca_7\ (SiO_4)_6\ (AlOH)_2;$$

and this fairly represents a good many other analyses of vesuvianite, but not all. Vesuvianite varies in composition, and its variations are rather troublesome to interpret, for they may signify either actual differences in the mineral itself, alterations, or impurities. The formula proposed by one of us[a] some years ago,

$$Al_2Ca_6\ (SiO_4)_5\ (AlOH),$$

is a good expression for the average composition of the species, but it does not fit the extremes.

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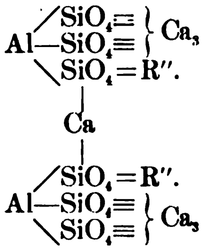
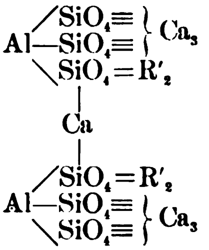
[a] Clarke, Bull. U. S. Geol. Survey No. 125, p. 25.

A careful study of over 40 analyses of vesuvianite leads us to suspect that it may be a mixture of several similar molecules, which we can represent by the subjoined expressions:

$$\begin{gathered}Al_2Ca_7(SiO_4)_6(AlOH)_2,\\ Al_2Ca_7(SiO_4)_6(AlO_2H_2)_4,\\ Al_2Ca_7(SiO_4)_6H_4,\\ Al_2Ca_7(SiO_4)_6Ca_2;\end{gathered}$$

the first and third compounds usually predominating. Structurally, and with due regard to the analogy between vesuvianite and other species, these formulæ may be written as follows:





About one-seventh of the calcium is replaced by magnesium, and in the variety wiluite the group BOH, analogous to AlOH, probably occurs.

Vesuvianite, then, is to be regarded as a basic orthosilicate, belonging to a group of compounds of which garnet is the normal salt. Epidote and the scapolites are other members of the same class, and their formulæ are all strikingly similar to one another. As products of contact metamorphism they originate under like conditions, and they all undergo alteration in much the same way, yielding similar or even identical derivatives. Garnet, vesuvianite, epidote, and the scapolites must be considered together if the problem of their constitution is to be properly solved.

# THE ACTION OF SILVER NITRATE AND THALLOUS NITRATE UPON CERTAIN NATURAL SILICATES.

By George Steiger.

The following investigation is practically a continuation of the experiments which were described by Clarke and Steiger in a recent bulletin of the United States Geological Survey,[a] although the new work is along somewhat different lines.

In the former research the alkalies and alkaline earth metals of certain silicates were replaced by ammonium, when the minerals were heated in sealed tubes with ammonium chloride. In the present work, silver nitrate and thallous nitrate have been substituted for ammonium chloride, the temperature employed being somewhat above their fusing points. As was anticipated, the alkalies and alkaline earth metals were replaced by the heavy metal, thus giving additional data relative to the constitution of the minerals.

The experiments were carried out as follows: The very finely ground material was mixed thoroughly with various amounts of nitrate by grinding in an agate mortar, the mixture was placed in a tube of Jena glass, and the tube was sealed. These tubes were then heated in a bomb furnace for the specified time and temperature, then opened and the contents leached with boiling water till the filtrate gave no test for nitrates. The residue was then dried and analyzed.

In a recent paper,[b] the writer gave the results of some preliminary work on analcite and chabazite.

Heumann[c] prepared a silver ultramarine by boiling the native mineral with a solution of silver nitrate, and A. H. Church[d] made a silver phillipsite by allowing a solution of silver nitrate to act on the natural mineral at ordinary temperatures for almost three years.

For the optical notes in the present paper, the writer is indebted to Messrs. W. T. Schaller and J. S. Diller.

The crystals in all cases were too small to permit a determination of their forms. A few optical observations were made, and with the exception of the isotropic compounds, it can be said that all were entirely crystalline.

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[a] Bull., U. S. Geol. Survey No. 207.

[b] Am. Jour. Sci., ser. 4, vol. 14, July, 1902, p. 31.

[c] Lieb. Ann., vol. 199 (1879), p. 253.

[d] Min. Mag., 1899, vol. 12, p. 9.

## ANALCITE.

The analcite upon which the following experiments were made was well crystallized material from Table Mountain, near Golden, Colo.

Three silver derivatives were prepared as follows:

A. Ammonium analcite was heated in a sealed tube with four times its weight of dry silver nitrate for 4 hours at 250°–290° C., then leached with boiling water and the residue dried on the water bath.

B. The natural analcite was heated in a sealed tube with four times its weight of dry silver nitrate for 4 hours at 250°–290° C., then leached and the residue dried on the water bath.

C. Same as B, but heated 6 instead of 4 hours.

D. Analysis of the analcite upon which the experiments were made.

Analysis gave the results shown below:

| | A. | B. | C. | D. |
|---|---|---|---|---|
| $SiO_2$ | 42.69 | 41.31 | 40.08 | 55.72 |
| $Al_2O_3$ | 18.22 | 16.44 | 16.29 | 23.06 |
| $Ag_2O$ | 32.01 | 37.45 | 36.91 | |
| CaO | | | | .17 |
| $Na_2O$ | .68 | .85 | .81 | 12.46 |
| $H_2O-$ | | | | .13 |
| $H_2O+$ | 6.09 | 4.29 | 5.86 | 8.26 |
| $NH_3$ | .69 | | | |
| | 100.38 | 100.34 | 99.95 | 99.80 |
| In leach water: | | | | |
| $Na_2O$ | .60 | 13.13 | 12.57 | |

The products were all perfectly white, showing that no free silver oxide contaminated the residue. Under the microscope a small amount of the material showed double refraction, but the larger portion of it was isotropic like the original mineral.

Residue A was boiled for 15 minutes with a 5 per cent solution of sodium carbonate, and only a trace of silica passed into solution. With nitric acid the silver analcite gives sandy silica, and does not gelatinize on evaporation.

A portion of residue B was heated on the water bath for 2½ hours with a 10 per cent solution of sodium hyposulphite and afterwards washed with water. In the residue 9.45 per cent of $Na_2O$ was found, showing that the silver salt had been converted back into the sodium compound, though not completely, the theoretical amount of $Na_2O$ required being 14.1 per cent. Whether a hydrated salt had been formed was not determined. After treatment in a similar manner with *potassium cyanide 10.04* per cent of $K_2O$ was found in the residue.

Considering the various amounts of alkali contained in the silver residues as unaltered analcite, deducting the equivalent amount and reducing the remainder to 100, the above analyses give the results shown below:[a]

| | A. | B. | C. | Theoretical Ag Al $Si_2O_6.H_2O$. |
|---|---|---|---|---|
| $SiO_2$ | 40.39 | 40.28 | 39.21 | 39.35 |
| $Al_2O_3$ | 17.27 | 15.92 | 15.90 | 16.72 |
| $Ag_2O$ | 36.65 | 39.78 | 39.16 | 38.03 |
| $H_2O$ | 5.69 | 4.02 | 5.73 | 5.90 |
| | 100.00 | 100.00 | 100.00 | 100.00 |

For comparison, there is given in the last column the percentage composition of theoretical silver analcite. Residue B shows a deficiency of water, possibly owing to the use of too high a temperature in drying the powder before analysis. Residue C gives the following molecular ratios:

$$26Ag_2O,\ 24Al_2O_3,\ 100SiO_2,\ 49H_2O,$$

agreeing well with the empirical formula

$$AgAlSi_2O_6.H_2O.$$

Two thallium derivatives were prepared from the same lot of analcite, as follows:

E. Natural analcite was heated in a sealed tube with four times its weight of thallous nitrate, twenty-four hours at 250°–290° C.

F. Same as E, except that the heating was continued for forty-eight hours. The composition of the two leached residues is given below.

| | E. | F. |
|---|---|---|
| $SiO_2$ | 34.40 | 32.66 |
| $Al_2O_3$ | 15.73 | 13.62 |
| $Tl_2O$ | 47.29 | 53.01 |
| $Na_2O$ | .60 | .12 |
| $H_2O-$ | .62 | .20 |
| $H_2O+$ | .76 | .40 |
| | 99.40 | 100.01 |

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[a] In these and all other deductions made in this work the theoretical molecule has been assumed, in this case, $NaAlSi_2O_6H_2O$.

Both of the preparations were perfectly white. The microscopic examination showed thallium analcite to be mainly isotropic, with a few crystals having double refraction, as was the case with the silver derivative. The thallium compound is easily decomposed by nitric acid, giving sandy silica.

Deducting from analysis F analcite equivalent to 0.12 per cent of $Na_2O$, and reducing to 100, the analysis becomes:

| | Residue F. | Theoretical $TlAlSi_2O_6$. |
|---|---|---|
| $SiO_2$ | 32.61 | 31.33 |
| $Al_2O_3$ | 13.60 | 13.32 |
| $Tl_2O$ | 53.79 | 55.35 |
| | 100.00 | 100.00 |

In the second column is given the calculated percentage composition of normal thallium analcite.

The molecular ratios of F are

$$24Tl_2O, 23Al_2O_3, 100SiO_2.$$

A closer agreement with theory could hardly be expected.

In the case of the silver compound, the one molecule of water of the original mineral has been retained, and it is essentially analcite with silver replacing sodium. It is not so with the thallium derivative, nor with the ammonium compound of the former investigation. In both of these derivatives the water of the original mineral was lost, the formula being that of leucite with thallium or ammonium replacing potassium, leucite being a mineral closely related to analcite.

One crude experiment was made to show the action of lead nitrate on this mineral. Natural analcite was heated in a sealed tube with a 10 per cent solution of lead nitrate for six hours at about 200° C. The composition of the washed and dried residue follows:

| | | Molecular ratios. |
|---|---|---|
| $SiO_2$ | 60.34 | 100.0 |
| $Al_2O_3$ | 17.55 | 17.1 |
| PbO | 14.75 | 6.1 |
| $Na_2O$ | 6.03 | 9.7 |
| $H_2O$ | undet. | .......... |
| | 98.67 | .......... |

Water was not determined, but, as will be seen from the summation, it is an unimportant constituent of the compound. Evidently the anhydrous salt has been formed, but incompletely, for only about half the sodium had been replaced.

The molecular ratios are as follows:

$$6PbO, 10Na_2O, 17Al_2O_3, 100SiO_2,$$

or, combining the monoxide bases,

$$16RO, 17Al_2O_3, 100SiO_2.$$

In this a large excess of silica appears.

## LEUCITE.

The specimen of leucite at hand was impure, and more careful work was therefore deferred until some future time, when it may be possible to prepare an artificial mineral which will be studied in detail.

One preparation of the impure leucite was made by heating the native mineral in a sealed tube with four times its weight of thallous nitrate for twenty-four hours at a temperature of 250°–290° C. The leached residue was dried on the water bath. The analysis of this residue follows.

| | Residue. | Molecular ratios. |
|---|---|---|
| $SiO_2$ | 36.00 | 100.0 |
| $Al_2O_3$ | 16.32 | 26.7 |
| $Tl_2O$ | 42.77 | 16.8 |
| $K_2O$ | 4.26 | 7.5 |
| $H_2O-$ | .10 | ......... |
| $H_2O+$ | .20 | 1.8 |
| | 99.65 | |

Combining potassium with thallium, the analysis gives the following:

$$24(TlK)_2O, 27Al_2O_3, 100SiO_2, 2H_2O.$$

This indicates that the same molecule has been formed as in the case of analcite.

## THOMSONITE.

The thomsonite used was well crystallized material from Table Mountain, Colorado. Only one derivative of this mineral was made. The thomsonite was heated in a sealed tube with four times its weight of dry silver nitrate, 20 hours, at 250°–290° C. The residue was dried *at about 40° C. Its* composition is given on the page following.

| | Natural thomsonite. | Silver thomsonite. |
|---|---|---|
| $SiO_2$ | 41.13 | 33.80 |
| $Al_2O_3$ | 29.58 | 23.21 |
| $Ag_2O$ combined | .......... | 23.50 |
| $Ag_2O$ free | .......... | 2.56 |
| CaO | 11.25 | 7.28 |
| $Na_2O$ | 5.31 | .71 |
| $H_2O-$ | 1.01 | .45 |
| $H_2O+$ | 12.12 | 8.11 |
| | 100.40 | 99.62 |
| In leach water: | | |
| CaO | .......... | 2.68 |
| $Na_2O$ | .......... | 5.04 |

The color of this residue was dark, showing that some silver oxide had been formed, so contaminating the salt. A determination of this free silver oxide was made by the following method, and it appears in the analysis as "$Ag_2O$ free."

A portion of the silver thomsonite was weighed out in a platinum boat and placed in a glass tube, which was connected with a weighed calcium chloride tube; low red heat was then applied and at the same time a current of dry air was passed over it. At this temperature silver oxide is decomposed, liberating the entire amount of oxygen. The loss of weight in the boat was water plus oxygen, the gain in the tube was water alone; the difference represented the oxygen of the free silver oxide.

The new compound dissolves easily in dilute nitric acid to a clear solution, which forms a perfect jelly on evaporation. Under the microscope the material proved to be entirely crystalline, showing parallel extinction.

Deducting free silver oxide, and also the water lost below 100°, and reducing to 100 per cent, the analysis gives the results tabulated below:

| | Silver thomsonite reduced. | Molecular ratios. |
|---|---|---|
| $SiO_2$ | 34.99 | 100.0 |
| $Al_2O_3$ | 24.02 | 40.3 |
| $Ag_2O$ | 24.32 | 18.0 |
| CaO | 7.54 | 23.2 |
| $Na_2O$ | 0.74 | 2.1 |
| $H_2O$ | 8.39 | 80.0 |
| | 100.00 | |

The reaction here is incomplete, but the thomsonite ratios have been fairly well preserved.

As the water of this mineral is usually considered to form no part of the chemical molecule, in the following formula it has been represented separately.

For this and the two minerals to follow, the formulæ proposed by F. W. Clarke have been adopted, regarding them as mixtures of salts of orthosilicic and trisilicic acids. The analysis gives the subjoined ratios:

$$18Ag_2O,\ 23CaO,\ 2Na_2O,\ 40Al_2O_3,\ 100SiO_2,\ 80H_2O.$$

Combining monoxide bases, we have

$$R''_{43}Al_{80}Si_{100}O_{363}.80H_2O;$$

which gives a ratio of orthosilicic to trisilicic acids of

$$73SiO_4 : 9Si_3O_8.$$

Substituting these, we have

$$R''_{43}Al_{80}(SiO_4)_{73}(Si_3O_8)_9.\ 80H_2O;$$

and combining the two acid radicles under the indiscriminate symbol X we have

$$R''_{43}Al_{80}X_{82}.80\ H_2O=1\text{-}2\text{-}2,\ 2H_2O.$$

The analysis of thomsonite reduced in the same way gives the formula

$$R''_{42}Al_{85}(SiO_4)_{76}(Si_3O_8)_8.\ 98H_2O,$$

or, condensed,

$$R''_{42}Al_{85}X_{84}.98H_2O=1\text{-}2\text{-}2,\ 2\tfrac{1}{2}H_2O.$$

## CHABAZITE.

The chabazite used in the following work was the well crystallized flesh-colored material from Wassons Bluff, Nova Scotia.

Two silver derivatives were made, as follows:

A. Ammonium chabazite was heated in an open tube with four times its weight of dry silver nitrate for six hours at 300° C.

B. Natural chabazite was heated in a sealed tube with four times its weight of dry silver nitrate for twenty-four hours at 250°-290° C.

C. Analysis of the original chabazite:

| | A. | B. | C. |
|---|---|---|---|
| $SiO_2$ | 38.84 | 34.95 | 50.78 |
| $Al_2O_3$ | 11.02 | 11.89 | 17.18 |
| $Fe_2O_3$ | | | .40 |
| $Ag_2O$ | 28.95 | 39.63 | |
| CaO | | | 7.84 |
| $Na_2O$ | | .40 | 1.28 |
| $K_2O$ | | | .73 |
| $H_2O-$ | | 2.38 | 5.22 |
| $H_2O+$ | | 4.40 | 16.63 |
| $N_2O_5$ | | 6.64 | |
| | | 100.29 | 100.06 |
| In the leach water: | | | |
| $Al_2O_3$ | | none | |
| CaO | | 7.75 | |
| $Na_2O$ | | 1.98 | |

This preparation was light in color, showing that no appreciable amount of free silver oxide could have been present. It was decomposed easily by nitric acid, giving sandy silica.

The microscopic examination showed the material to be entirely crystalline and uniaxial.

In this, and as will be shown later in the thallium derivatives also, important amounts of $N_2O_5$ have been retained. This is not due to incomplete washing, but the nitrate is firmly held within the molecule. This same mineral gave, with ammonium chloride, a compound containing chlorine. What the presence of these small amounts of the acid radicles may mean is not known, but in the following reductions $N_2O_5$ will be considered as combined directly with silver or thallium; the proper amount will be subtracted from the analysis.

Deducting water below 100° and also silver nitrate, analysis B becomes:

| | Residue B. | Molecular ratios. |
|---|---|---|
| $SiO_2$ | 45.30 | 100.0 |
| $Al_2O_3$ | 15.41 | 20.0 |
| $Ag_2O$ | 33.07 | 19.0 |
| $Na_2O$ | .52 | 1.1 |
| $H_2O$ | 5.70 | 42.0 |
| | 100.00 | |

This gives the following molecular ratios:

$$19Ag_2O,\ 1Na_2O,\ 20Al_2O_3,\ 100SiO_2,\ 42H_2O.$$

Combining sodium with silver, we have

$$R''_{20}Al_{40}Si_{100}O_{280}.42H_2O.$$

Hence the ratio between orthosilicic and trisilicic acids becomes

$$10SiO_4.30Si_3O_8;$$

and combining them under the symbol X, the formula is

$$R''_{20}Al_{40}X_{40}.42H_2O = 1\text{-}2\text{-}2,\ 2H_2O.$$

Reducing the same way the analysis for chabazite, we have

$$R''_{20}Al_{40}X_{40}.109H_2O = 1\text{-}2\text{-}2,\ 5\tfrac{1}{2}H_2O.$$

The two formulæ are

| | |
|---|---|
| Native chabazite | $R''_{20}Al_{40}(SiO_4)_{10}(Si_3O_8)_{30}.109H_2O.$ |
| Silver chabazite | $R''_{20}Al_{40}(SiO_4)_{10}(Si_3O_8)_{30}.42H_2O.$ |

Here calcium has been replaced by silver with a loss of three and one-half molecules of water.

The thallium derivatives were prepared with the same lot of chabazite. Two preparations were made, as follows:

D. Chabazite was heated in a sealed tube with four times its weight of thallous nitrate, forty-eight hours at 250°–290° C.

E. Treated the same as D, but heated for sixty-seven hours.

| | D. | | | E. | |
|---|---|---|---|---|---|
| | Residue found. | Residue reduced. | Molecular ratios. | Residue found. | Residue reduced. |
| $SiO_2$ | 28.92 | 36.31 | 100.0 | 29.44 | 34.44 |
| $Al_2O_3$ | 10.75 | 13.50 | 21.8 | 11.11 | 13.00 |
| $Tl_2O$ | 51.58 | 47.39 | 18.4 | 52.11 | 49.61 |
| CaO | .65 | .81 | 2.3 | .64 | .75 |
| $Na_2O$ | .28 | .35 | 1.0 | .35 | .41 |
| $H_2O$ | 1.30 | 1.64 | 14.7 | 1.53 | 1.79 |
| $H_2O-$ | 2.85 | ........ | ........ | 2.60 | ........ |
| $N_2O_5$ | 3.54 | ........ | ........ | 2.50 | ........ |
| | 99.87 | 100.00 | ........ | 100.28 | 100.00 |
| In the leach water: | | | | | |
| $Al_2O_3$ | ........ | ........ | ........ | None. | ........ |
| CaO | ........ | ........ | ........ | 7.09 | ........ |
| $Na_2O$ | 1.54 | ........ | ........ | 1.58 | ........ |

Both of these residues were perfectly white and entirely crystalline.

Thallium chabazite is easily decomposed with nitric acid, yielding sandy silica.

Derivative D gives the following ratios:

$$18Tl_2O,\ 2CaO,\ 1Na_2O,\ 22Al_2O_3,\ 100SiO_2,\ 15H_2O;$$

or, combining monoxide bases,

$$R''_{21}Al_{44}Si_{100}O_{287}.15H_2O.$$

The ratio of the two acid radicles is

$$16SiO_4 : 28Si_3O_8;$$

and, inserting these in the formula, we have

$$R''_{21}Al_{44}(SiO_4)_{16}(Si_3O_8)_{28}.15H_2O;$$

or, condensed,

$$R''_{21}Al_{44}X_{44}.15H_2O=1-2-2+aq.$$

A rough experiment was made by heating silver chabazite in a sealed tube with a 10 per cent solution of lead nitrate. The residue after washing and drying contained 5.59 per cent $Ag_2O$ and 20.24 per cent PbO.

## STILBITE.

The specimen of stilbite used came from Wassons Bluff, Nova Scotia. The action of silver nitrate upon this mineral was not investigated.

One thallium derivative was made by heating the stilbite in a sealed tube with four times its weight of thallous nitrate sixty-five hours at 230°-280° C. The residue was perfectly white, entirely crystalline, and biaxial, and it gave sandy silica with nitric acid. The analysis follows:

| | Stilbite. | Residue. | Molecular ratios. |
|---|---|---|---|
| $SiO_2$ | 55.41 | 36.75 | 100.0 |
| $Al_2O_3$ | 16.85 | 11.74 | 18.8 |
| $Fe_2O_3$ | .18 | | |
| $Tl_2O$ | | 42.94 | 16.5 |
| CaO | 7.78 | .68 | 2.0 |
| $Na_2O$ | 1.23 | .15 | .3 |
| $H_2O-$ | 3.60 | 5.48 | |
| $H_2O+$ | 15.41 | 2.29 | 20.8 |
| | 100.46 | 100.03 | |

Omitting water lost below 100°, we have the following ratios:

$$16\ Tl_2O,\ 2\ CaO,\ 1\ Na_2O,\ 19\ Al_2O_3,\ 100\ SiO_2,\ 20\ H_2O;$$

and, combining monoxide bases,

$$R''_{19}Al_{38}Si_{100}O_{276},\ 20\ H_2O;$$

which gives a ratio between the two acid radicles of

$$7\ SiO_4 : 31\ Si_3O_8.$$

Inserting these in the formula, it becomes

$$R''_{19}Al_{38}(SiO_4)_7(Si_3O_8)_{31}.20\ H_2O,$$

which condenses to

$$R''_{19}Al_{38}X_{38}.20\ H_2O.$$

Reducing the analysis of stilbite in the same way, we get

$$R''_{17}Al_{36}(SiO_4)_4(Si_3O_8)_{32}.93\ H_2O=1-2-2,\ 5\ H_2O.$$

From the last three minerals we get the following formulæ:

| | |
|---|---|
| Native thomsonite | $R''_{42}Al_{85}(SiO_4)_{76}(Si_3O_8)_8.98H_2O=1-2-2,2\frac{1}{2}H_2O.$ |
| Silver thomsonite | $R''_{43}Al_{80}(SiO_4)_{73}(Si_3O_8)_9.80H_2O=1-2-2,2H_2O.$ |
| Native chabazite | $R''_{20}Al_{40}(SiO_4)_{10}(Si_3O_8)_{30}.109H_2O=1-2-2,5\frac{1}{2}H_2O.$ |
| Silver chabazite | $R''_{20}Al_{40}(SiO_4)_{10}(Si_3O_8)_{30}.42H_2O=1-2-2,2H_2O.$ |
| Thallium chabazite | $R''_{21}Al_{44}(SiO_4)_{16}(Si_3O_8)_{28}.15H_2O=1-2-2,Aq.$ |
| Native stilbite | $R''_{17}Al_{36}(SiO_4)_4(Si_3O_8)_{32}.93H_2O=1-2-2,5H_2O.$ |
| Thallium stilbite | $R''_{19}Al_{38}(SiO_4)_7(Si_3O_8)_{31}.20H_2O=1-2-2,Aq.$ |

The similarity of the foregoing formulæ is plainly evident. Leaving water out of consideration, and except in the single case of silver thomsonite, the ratios agree closely with the ratios of 1-2-2 for the monoxide base, aluminum, and silicic acid. It will be seen that with the two silver derivatives, compounds retaining two molecules of water were formed, although in the case of thomsonite a mineral hydrated with 2½ molecules of water was taken, while the chabazite contained 5½.

## NATROLITE.

The natrolite which was studied came from Bergen Hill, N. J. Two silver derivatives were made, as follows:

A. Natrolite was heated in a sealed tube with a 10 per cent solution of silver nitrate, eighteen hours at 120° C.

B. Natrolite was heated in a sealed tube with four times its weight of dry silver nitrate, twenty-two hours at 250°-290° C.

| | A. | B. | B. reduced. | Molecular ratios. |
|---|---|---|---|---|
| $SiO_2$ | 40.45 | 34.70 | 36.25 | 100.0 |
| $Al_2O_3$ | 10.24 | 18.54 | 19.36 | 31.4 |
| $Ag_2O$ combined | 20.48 | 32.30 | 33.73 | 24.0 |
| $Ag_2O$ free | | 3.91 | .......... | .......... |
| CaO | .93 | 2.50 | 2.61 | 7.6 |
| $Na_2O$ | 7.84 | .39 | .41 | 1.0 |
| $H_2O$ | .......... | 7.32 | 7.64 | 70.2 |
| | .......... | 99.66 | 100.00 | .......... |

Both preparations were very dark, showing that a quantity of free silver oxide had been deposited. The compound formed is easily decomposed by warm dilute nitric acid, giving a perfectly clear solution which gelatinizes upon evaporation. Under the microscope, it showed perfect crystallization.

Neglecting free silver oxide, residue B gives the following formula:

$$24Ag_2O, 8CaO, 1Na_2O, 31Al_2O_3, 100SiO_2, 70H_2O,$$

or, combining monoxide bases,

$$R'_{66}Al_{62}Si_{100}O_{326}, 70H_2O,$$

which corresponds roughly to the formula $R_2Al_2Si_3O_{10}.\ 2H_2O$. The ratios are not so good as could be desired, but are close enough to show that this compound has probably been formed.

One thallium derivative was made by heating natrolite in a sealed tube with six times its weight of thallous nitrate for forty-eight hours, at 250°-290° C. The analysis gave the following results:

| | Residue. | Molecular ratios. |
|---|---|---|
| $SiO_2$ | 26.10 | 100.0 |
| $Al_2O_3$ | 15.59 | 35.1 |
| $Tl_2O$ | 55.64 | 30.1 |
| CaO | .67 | 2.8 |
| $Na_2O$ | .34 | 1.1 |
| $H_2O$ | .51 | .......... |
| $H_2O+$ | .66 | 8.5 |
| | 99.51 | .......... |

This compound is perfectly white. It is easily dissolved by dilute nitric acid, giving a perfect jelly on evaporation.

The optical observations were: Double refraction low, cleavage vertical, extinction parallel, probably biaxial. It was entirely crystalline. The molecular proportions are as follows:

$$30Tl_2O,\ 3CaO,\ 1Na_2O,\ 35Al_2O_3,\ 100SiO_2,\ 9H_2O;$$

or, combining monoxide bases,

$$R'_{68}Al_{70}Si_{100}O_{339}.\ 9H_2O,$$

which corresponds closely to the formula $R_2Al_2Si_3O_{10}$.

## SCOLECITE.

The scolecite used came from Whale Cove, on the island of Grand Manan, New Brunswick.

One derivative was made by heating the scolecite in a sealed tube with four times its weight of silver nitrate six hours at 250°–280° C. The residue was dried at about 35° C.

| | Scolecite. | Residue. | Molecular ratios. |
|---|---|---|---|
| $SiO_2$ | 45.86 | 37.28 | 100.0 |
| $Al_2O_3$ | 25.78 | 21.54 | 34.0 |
| $Ag_2O$ combined | ......... | 20.21 | 14.0 |
| $Ag_2O$ free | ..... .... | 6.67 | .......... |
| CaO | 13.92 | 5.40 | 15.5 |
| $Na_2O$ | .41 | .44 | 1.1 |
| $H_2O-$ | .40 | .46 | .......... |
| $H_2O+$ | 13.65 | 7.92 | 70.8 |
| | 100.02 | 99.92 | .......... |

This residue was dark brown, owing to the silver oxide it contained. On heating over the low flame of the Bunsen burner in order to reduce the silver oxide to metallic silver it turned perfectly white.

On attempting to mount this preparation for microscopic examination the balsam became very dark, which prevented any optical observations from being made.

The molecular ratios for this derivative are:

$$14Ag_2O,\ 16CaO,\ 1Na_2O,\ 34Al_2O_3,\ 100SiO_2,\ 71H_2O,$$

or, uniting monoxide bases,

$$R'_{62}Al_{68}Si_{100}O_{333},\ 71H_2O.$$

These proportions agree fairly well with the formula

$$R'_2Al_2Si_3O_{10}, +aq.$$

The reaction in this case was incomplete, only a little over one-half the calcium and none of the sodium being replaced, but the experiment shows that the desired molecule was partially formed.

## MESOLITE.

The mesolite studied was a fine specimen of crystalline material from Österö, in the Faroe Islands.

One thallium derivative was prepared by heating the mesolite in a sealed tube with four times its weight of dry thallous nitrate for eighty hours at 240°-280° C.

Its analysis and that of the mesolite follow:

| | Mesolite. | Residue. | Molecular ratios. |
|---|---|---|---|
| $SiO_2$ | 45.97 | 29.11 | 100.0 |
| $Al_2O_3$ | 25.98 | 16.94 | 34.2 |
| $Tl_2O$ | | 44.75 | 21.7 |
| $CaO$ | 9.69 | 2.90 | 10.8 |
| $Na_2O$ | 4.79 | .26 | .8 |
| $H_2O-$ | 1.34 | 3.32 | |
| $H_2O+$ | 12.04 | 3.06 | 35.0 |
| | 99.81 | 100.34 | |
| In the leach water: | | | |
| $Al_2O_3$ | | None. | |
| $CaO$ | | 5.50 | |
| $Na_2O$ | | 4.42 | |

The leached residue was perfectly white and entirely crystalline, with parallel extinction. It gave a perfectly clear solution with acid, which gelatinized on slight evaporation.

The molecular ratios are as follows:

$$22Tl_2O,11CaO,1Na_2O,34Al_2O_3,100SiO_2,35H_2O.$$

Combining monoxide bases, we have

$$R'_{68}Al_{68}Si_{100}O_{336}.35H_2O,$$

agreeing almost perfectly with the formula

$$R'_2Al_2Si_3O_{10}.H_2O.$$

These three minerals and their derivatives give the following comparison, when all the monad and dyad bases are united under R'':

| | |
|---|---|
| Native natrolite | $R''_{36}Al_{66}Si_{100}O_{335}$. 72 $H_2O$ |
| Silver natrolite | $R''_{33}Al_{62}Si_{100}O_{326}$. 70 $H_2O$ |
| Thallium natrolite | $R''_{34}Al_{70}Si_{100}O_{339}$. 9 $H_2O$ |
| Native scolecite | $R''_{34}Al_{66}Si_{100}O_{333}$. 99 $H_2O$ |
| Silver scolecite | $R''_{31}Al_{68}Si_{100}O_{333}$. 71 $H_2O$ |
| Native mesolite | $R''_{33}Al_{66}Si_{100}O_{332}$. 87 $H_2O$ |
| Thallium mesolite | $R''_{34}Al_{68}Si_{100}O_{336}$. 35 $H_2O$ |

Since, as others have shown, the water in these three minerals plays no part in the chemical molecule, it has been placed as a separate constituent in the formulæ.

The ratios found are not so sharp as could be desired, nor have the reactions been complete, but they indicate that the three minerals are all derived from the trisilicic acid $H_8Si_3O_{10}$.

Deducting from each of the above formulæ the water contained in the undecomposed mineral, which latter is represented by the amount of alkali retained by the derivative, we have the water contents:

| | |
|---|---|
| Silver natrolite | 70 $H_2O$ less 18 $H_2O$ equals 52 $H_2O$ |
| Thallium natrolite | 9 $H_2O$ less 8 $H_2O$ equals 1 $H_2O$ |
| Silver scolecite | 71 $H_2O$ less 50 $H_2O$ equals 21 $H_2O$ |
| Thallium mesolite | 35 $H_2O$ less 29 $H_2O$ equals 6 $H_2O$ |

We have here, as was the case with the thomsonite, chabazite, and stilbite derivatives, silver compounds strongly hydrated, while the thallium salts are practically anhydrous.

## PECTOLITE.

The pectolite came from Bergen Hill, N. J. It contained a small amount of calcium carbonate as an impurity.

A single silver derivative was made by heating the pectolite in a sealed tube with four times its weight of dry silver nitrate, for twenty-two hours at 240°-280° C. The residue was dried on the water bath. On analysis the following results, stated below, were obtained.

| | Pectolite. | Residue. | Molecular ratios. |
|---|---|---|---|
| $SiO_2$ | 53.34 | 39.05 | 100.0 |
| $Al_2O_3$ | .33 | .38 | ........... |
| $Ag_2O$ | .......... | 32.57 | 21.5 |
| CaO | 33.23 | 18.13 | 49.8 |
| MnO | .45 | .......... | ........... |
| $Na_2O$ | 9.11 | 5.99 | 14.9 |
| $H_2O-$ | .27 | .......... | ........... |
| $H_2O+$ | 2.70 | 4.12 | 35.2 |
| $CO_2$ | .67 | .......... | ........... |
| | 100.10 | 100.24 | .......... |

The derivative was dark, but on heating over the bunsen burner the color disappeared, and on treating the ignited residue with nitric acid red fumes were given off, showing that a quantity of free silver oxide was present. This was not determined, but is contained in the figures given for $Ag_2O$. The silver oxide reported is therefore several per cent too high.

The compound is easily decomposed by warm dilute nitric acid, with the separation of sandy silica. A complex breaking up of the molecule has taken place and no simple ratios appear.

## ELÆOLITE.

Elæolite from Litchfield, Me., was heated in the usual way with thallous nitrate. An incomplete analysis of the leached residue gave the following results:

| | |
|---|---|
| Insoluble in nitric acid | 46.34 |
| $Al_2O_3$ | 27.90 |
| $Tl_2O$ | 8.50 |
| $Na_2O+K_2O$ | 15.40 |
| Water, etc | undet. |

In the leach water was found 2.13 per cent of sodium and potassium oxides. The analysis shows that elæolite is but slightly attacked by treatment with thallous nitrate.

## ÆGIRITE.

Ægirite from Magnet Cove, Arkansas, was heated in a sealed tube with dry silver nitrate. The following results were obtained on analysis of the leached residue:

| | |
|---|---|
| Insoluble in nitric acid | 91.22 |
| $Al_2O_3+Fe_2O_3$ | 1.18 |
| CaO | small |
| $Ag_2O$ | 4.12 |
| $Na_2O$ | .53 |
| Water, etc | undet. |

This mineral has evidently been but little altered by the action of silver nitrate.

# DUMORTIERITE.

By WALDEMAR T. SCHALLER.

## INTRODUCTION.

Dumortierite was discovered by Gonnard[a] in November, 1879, at Chaponost, near Lyons, France, and was made the subject of a brief note by E. Bertrand in 1880. The following year, Gonnard named the mineral after Eugene Dumortier, the eminent French paleontologist.

The mineral was known to be new from its optical properties before its chemical composition was determined, its beautiful pleochroism being especially marked. Damour soon after made an analysis of the mineral, from which he obtained the formula 4 $Al_2O_3$, 3 $SiO_2$.

Specimens of this mineral from New York, previously taken for indicolite, were studied by Diller and Riggs and shown to be different from tourmaline. At about the same time dumortierite from Arizona became known, and analyses of the mineral from both localities were made. These analyses all showed the presence of boric acid in varying amounts, and also a small quantity of water. An occurrence of dumortierite in Norway was afterward noted, and still later several localities in Germany were mentioned.

In 1902, Ford published three analyses of dumortierite, one on material from New York, a second on material from Arizona, and a third on material from a new locality in San Diego County, Cal. In the same paper he mentions a second new locality, the fourth one in this country, in Skamania County, Wash. He showed by his analyses that dumortierite contains boric acid and water in nearly constant amount.

A careful search for crystals was made in a large quantity of the California mineral, which resulted in the finding of two small, imperfect but measurable examples. On looking over the Arizona material in the National Museum a third crystal was found. Three crystals were also obtained among specimens from New York. These were carefully measured and some crystallographic data were thus obtained.

The mineral is usually sparingly present in small fibrous masses scattered through the rock, making it difficult to obtain any considerable quantity of it in a pure condition. At the California locality the mineral occurs in large, pure masses, very favorable for chemical investigation. From an analysis made by the writer on the California material, a formula was deduced in which the boric acid and water have a certain fixed value. The other analyses are discussed, and it is shown that they all agree fairly well with the new formula proposed by the writer. An analysis of the Washington dumortierite is also given. Ford tested the French material qualitatively for boric acid and obtained a strong positive reaction.

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[a] For references see Literature, pp. 119-120, this bulletin.

While a more or less complete study of the mineral has been made, it is fully realized that in no sense can the work be called complete. The material for crystallographic study was poor, and it is very much hoped that better crystals will at some future time be found. The determinations of optical constants are very meager, for the usual condition of occurrence of the mineral is not favorable to such study. Only a few analyses are available for any discussion of the composition of the mineral, and though dumortierite does not contain a large number of constituents, many more analyses—but only extremely accurate and reliable ones—will be gladly welcomed.

It was found that the California material differed somewhat in its properties from normal dumortierite. The color is lavender instead of blue, the pleochroism is colorless to red purple instead of blue, and the mineral contains 1½ per cent titanium oxide. It is of especial interest to correlate these differences, as it adds another good example to that class of silicates which contain a small amount of titanium and (seemingly thereby) acquire a purple pleochroism.

## OCCURRENCES.

### FOREIGN.

#### FRANCE.

Dumortierite[a] is found at several places in the vicinity of Lyons (Rhone), where it occurs in a pegmatite gneiss. At Beaunan, on the road from Oullins to Chaponost, it is found in a small quarry in gneiss in slender blue fibers in the midst of veinlets or lenses of feldspar and pegmatite. At Brignais it occurs in very small fibers disseminated throughout a white pegmatite, associated with large black tourmalines, garnets, muscovite, and cordierite, the latter being largely altered to mica. In some cases bluish tourmaline and dumortierite have the vertical axis in common. Pseudomorphs of mica after dumortierite are frequent.

The mineral has a good cleavage parallel to *a* {100} and an imperfect prismatic one. There are also "planes of separation" parallel to the base. Twins, analogous to those of aragonite, are frequent. The color is blue, sometimes nearly black. Inclusions are not prominent, though bubbles and rutile have been noticed.

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[a] Lacroix, A., Minéralogie de la France, vol. 1, 1893-1895, pp. 15-19.

The axial plane is parallel to *b* {010}. Bx$_a$ (negative) perpendicular to *c* {001}. In the twins the axial planes of the two parts make an angle of about 60°. $\beta$ is about 1.65, and the birefringence is about .010. The axial angle (2 V$_{Na}$) is near 35° to 40°. The dispersion is energetic $\rho<v$. Pleochroism is very strong; $\mathfrak{b} = \mathfrak{c}$, colorless or very pale blue; $\mathfrak{a}$, cobalt blue or violet. Some of the specimens from Brignais are of a rose-salmon color, a little redder than andalusite. The maximum absorption is (contrary to that of tourmaline) in the direction of elongation (parallel to $\acute{c}$). The presence of idiophanous figures is also noted. According to Damour, the blue color of the mineral may perhaps be due to traces of $Ti_2O_3$, the presence of which in the French mineral has, however, not been demonstrated. It is noticed that the small fibers of dumortierite are often inclosed in a white mica, which is gradually replacing the original mineral.

### GERMANY AND AUSTRIA.

The mineral is found near Schmiedeberg, Silesia,[a] in the pegmatite of the lower Val Donbastone, in long fibers of a green or greenish-blue color. These fibers have a thickness of 1 mm. and reach a maximum length of 50 mm. They are often bent, and inclose pieces of quartz. On one fiber a prism angle of 69° 6′ was measured. Prismatic cleavage, imperfect, was noted. The pleochroism is very strong: $\mathfrak{c} = \mathfrak{b}$, colorless; $\mathfrak{a}$, pistachio-green. Cross sections show the emergence of an acute bisectrix (negative), the axial angle being somewhat smaller than that of muscovite. With a crystal refractometer the following values for the indices of refraction were obtained:

$$\begin{aligned}\alpha&=1.678\\ \beta&=1.686\\ \gamma&=1.689\\ \gamma-\alpha&=\ .011\end{aligned}$$

An analysis of impure material was also made.

In a clayey residue, the mineral has been found at Imligan, near Chodan, and Schobrowitz, near Carlsbad, in Bohemia, and in clayey sandstone at Oberbris, near Pilsen, in Bohemia. These residues are probably formed from decomposing pegmatite, from which the dumortierite originally came. The mineral shows a strong pleochroism; $\mathfrak{a}$, deep carmine to smalt blue and, in places where partly decomposed, dark olive to red brown; $\mathfrak{b}$, light gray olive to colorless. The (negative) acute bisectrix is parallel to the elongation. The obtuse bisectrix is perpendicular to the cleavage plates ($a$={100}). The associated minerals are tourmaline, quartz, feldspar, muscovite, biotite, chrysoberyl, apatite, garnet, and chlorite.

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[a] See Literature, pp. 119-120.

### NORWAY.

The mineral is found at Tvedestrand in minute fibers in cordierite gneiss. The pleochroism is as follows:

$\mathfrak{c} = \mathfrak{b}$, pale yellow, almost colorless.
$\mathfrak{a}$, intense cobalt blue.

Cross sections show a prismatic angle of about 60°. The birefringence (.010) is feeble. Pleochroic halos (in yellow) are very characteristic. The axial angle (2V) is about 35°, and the dispersion strong $\rho > v$. The associated minerals are cordierite and sillimanite.

Romberg has described a mineral from Argentina (2.51 per cent $B_2O_3$) which seems to resemble the California dumortierite. The color is lilac and the pleochroism similar to that from California.

## UNITED STATES.

### NEW YORK.

The mineral occurs in that part of New York City known as Harlem. It has been sparingly found as acicular crystals "aggregated in fasces or tufts resembling clippings of hair" in a lode of red granite extending southeastward from Fourth avenue and One hundred and twenty-third street to Madison avenue at One hundred and sixteenth street. At One hundred and seventy-first street and Fort Washington avenue it was found in several pockets in a vein of coarse pegmatite in mica-schist. The vein is about 3 feet wide and the pockets extended for about 80 feet. The body of the vein is granular gray quartz, orthoclase, and flaky muscovite. The dumortierite occurs chiefly in the orthoclase, though it has also been found as long, filiform inclusions in the muscovite, singly and radiating from centers. It has also been noticed at Kips Bay, near the upper end of Riverside Park, in white oligoclase, and at Tenth avenue and One hundred and thirtieth street.

The associated minerals are orthoclase, quartz, muscovite, xenotime, monazite, tourmaline, zircon, torbernite (?), autunite (?), apatite, garnet, and andalusite.

A brief study of several slides of the New York City dumortierite showed that the mineral occurs almost entirely in the orthoclase. Very rarely is it in the quartz or muscovite. It is usually in fibrous forms, with ragged outlines, and in one particular section an orthoclase crystal was filled with minute threads of dumortierite, arranged more or less parallel. In these slides andalusite also was noticed.

Mr. Frederick Braun, of Brooklyn, N. Y., very kindly lent the writer his private collection of New York dumortierite, which probably contains the finest dumortierite specimens ever found in that locality. About 25 specimens are shown, the best one containing a prismatic group of fine blue dumortierite fibers 5 cm. long and from one-half to 1 cm. thick. The rock is a pegmatitic gneiss, and consists chiefly of quartz and feldspar, with smaller quantities of biotite, muscovite, black tourmaline, dumortierite, and garnets. The dumortierite occurs in long prisms, often bent, and again broken, the space between being filled with the gneiss rock. The basal cross fracture is common and the color is often changed to a dull gray. At times the fibers are seemingly intergrown with the tourmaline in parallel position. The three crystals described later were obtained from these specimens.

As additional localities, Mr. Braun gives: One hundred and first street and Lexington avenue, New York; One hundred and eighteenth, One hundred and twentieth, One hundred and twenty-second streets and Madison avenue; One hundred and thirty-eighth, One hundred and thirty-ninth, One hundred and fortieth, One hundred and forty-ninth streets and Mott avenue; One hundred and thirty-third, One hundred and thirty-fifth, One hundred and forty-ninth streets and Hudson River Railroad; One hundred and forty-ninth street and Sheerier place; One hundred and seventy-first street and Boulevard.

The mineral possesses a strong pleochroism, ranging from colorless to deep blue, this color being always in the direction of elongation. The extinction is parallel and the mineral orthorhombic. Cleavage parallel to the macropinacoid is perfect, these cleavage plates showing the emergence of an obtuse bisectrix. An imperfect cleavage parallel to the base is also noticed, and basal sections show an imperfect prismatic cleavage. Such sections show the emergence of an acute bisectrix. The mineral is negative. The axial plane is parallel to the brachypinacoid and the orientation is $a=\mathfrak{c}$, $b=\mathfrak{b}$, $c=\mathfrak{a}$. The pleochroism is $\mathfrak{c}=\mathfrak{b}$, colorless; $\mathfrak{a}$, deep blue. Absorption, $\mathfrak{a}>\mathfrak{b}=\mathfrak{c}$. Some sections show polysynthetic twinning lamellæ, but their relation could not be made out.

## ARIZONA.

At Clip, Ariz., dumortierite occurs in a quartz rock which has not been found in place (Doctor Hillebrand), but only as loose bowlders. Dumortierite and quartz form the principal mineral constituents, kyanite, magnetite, and muscovite being present in subordinate amounts. The rock is fine grained and has a blue color due to the dumortierite.

Under the microscope, a section of the rock showed numerous small lath-shaped prisms of dumortierite placed in every direction in a mass of allotriomorphic quartz, with several large blades of muscovite and abundant magnetite, and a little apatite and rutile. Feldspar seems to be entirely absent.

The dumortierite has the usual colorless to blue pleochroism and presents normal properties. In the slide studied it is very fresh and shows no signs of alteration, the muscovite present being primary.

The quartz grains are (relatively) fairly large and are not crushed like those of the California rock. The rock seems to have undergone no dynamic change after its formation.

The pale-green kyanite has been analyzed by Dr. W. F. Hillebrand with the results given below:

| | |
|---|---|
| $SiO_2$ | 36.30 |
| $Al_2O_3(+TiO_2)$ | 62.51 |
| $Fe_2O_3$ | .70 |
| FeO | undet. |
| CuO | tr. |
| Ign | .40 |
| | 99.91 |
| Specific gravity | 3.656 at 18.5 |

### CALIFORNIA.

This occurrence was first mentioned by Ford, and an analysis of the dumortierite was given by him. The writer made a brief visit to the locality in the summer of 1903 and collected a large quantity of material, which permitted a rather complete study of the mineral. The writer's thanks are due to Mr. John A. Thoman, of San Diego, the owner of the property, for permission to visit the interesting locality and for the privilege of collecting specimens.

The mineral occurs in a dike a few miles east of Dehesa, San Diego County, and is not far from the orbicular gabbro described by Professor Lawson.[a] The general country rock of the region is granite, in which occur large masses of gabbro, of which the orbicular rock is a peculiar phase. Both the gabbro and the granite contain many dikes, and it is in one of these dikes, in decomposed biotite-granite, that the dumortierite occurs. This dike has a length of about 1,000 feet and a thickness of about 30 or 40 feet, strikes S. 70° E., and has a dip of about 70° N. 20° E.

The rock consists chiefly of quartz and either sillimanite or dumortierite. Feldspar is entirely absent from the dike. The dike may be divided into an upper and lower part, according to the character of the rock. The upper part is fine grained and consists of quartz and sillimanite; the lower part is coarse grained and consists of quartz and dumortierite.

The upper part, forming about one-half the dike, is a fine-grained white saccharoidal rock, sometimes of a grayish color, and never contains any dumortierite. It possesses a more or less schistose structure and cleaves off into blocks parallel to the dip. A little muscovite is sometimes present, small crystals of titanium oxide are abundant, and occasionally a little pyrite is seen.

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[a] Bull. Dept. Geol. Univ. Cal., vol. 3, 1904, pp. 383-396.

Microscopically the rock is composed of allotriomorphic quartz, sillimanite prisms arranged in parallel bands (schistose structure), a little muscovite, probably secondary, and accessory titanite, titanium oxide, rutile (?), pyrite, apatite, zircon, and corundum.

The sillimanite occurs both as short and long prisms. The former are usually grouped together in parallel bands which may run across the entire slide or may pinch out at both ends, forming a very flat lens. The prisms are short and stout, usually more or less ragged in outline, with many transverse fractures. The several parts of a single prism thus broken are at times considerably displaced. Again, an entire row of prisms may be broken and displaced. The displacement is always in the direction of elongation of the prism. The ends are usually more or less rounded, and macropinacoidal cleavage is sometimes well developed. Some nearly square cross sections show diagonal cleavage. Titanium oxide and zircon are at times inclosed in the prisms. These prisms occasionally show slight alterations and at times seem to have been entirely removed, forming either an opaque mass or leaving just the outlines of former prisms behind.

The long, slender prisms are placed at all angles and are not grouped in bands as the others are. The edges are sharp and the ends are often terminated by two faces, forming a spear. These prisms are usually smaller than the others and decrease in thickness until they are so narrow as to become entirely opaque. These straight hair-like inclusions were at first taken for rutile or magnetite, but it is thought they are merely sillimanite prisms of extreme thinness.

Optically the two forms are identical, and are to be classified as sillimanite. The relief is distinct and birefringence is rather strong. The direction of elongation is always an axis of minimum elasticity. The crystals are nonpleochroic and always extinguish parallel. Pleochroic halos were not observed.

The quartz is allotriomorphic and is usually much cracked. In general the principal cracks are parallel to the bands of sillimanite, while cracks perpendicular to these bands are also numerous. Many transverse irregular cracks also occur. Some of the quartz grains seem to have been slightly displaced, and the cracks thus formed been filled with secondary quartz. Sometimes the bands of sillimanite have caused "augen" to form. In places the quartz shows large liquid inclusions, and strings of small inclusions, though not common, sometimes occur. Besides the inclusions mentioned and the accessory minerals, the quartz contains in places a large amount of very fine opaque dust, possibly magnetite. Such sections of the quartz as remain dark under crossed nicols show a uniaxial cross with a positive sign in convergent light.

Of the accessory materials, titanium oxide, probably rutile, occurs in the greatest abundance. A few grains are probably titanite, but the majority of the small sharply defined crystals are probably rutile. They are at times slightly dichroic (yellow to brown), and occur in equidiametral crystals. Some of the crystals in the bands of sillimanite have been crushed and lengthened out. In one case a crystal of corundum was inclosed in rutile. Zircon is also widely distributed in irregular grains, usually of minute size. Occasionally a large piece has roughly the usual crystal outline, but seems to have been more or less corroded. One large crystal inclosed several small rutiles. The order of crystallization of the minerals seems to have been as follows: Corundum, rutile (and titanite), zircon, sillimanite, and quartz. It may be that there are two generations of sillimanite. Those long, slender sharp crystals in the quartz may have formed after the crystals in the bands.

An analysis of the rock was made by the writer, giving the results tabulated below:

| | |
|---|---|
| $SiO_2$ | 75.54 |
| $Al_2O_3$ | 18.65 |
| $Fe_2O_3$ | .35 |
| FeO | .06 |
| MgO | none |
| CaO | .03 |
| Alk | none |
| $H_2O-$ | 1.10 |
| $H_2O+$ | 3.67 |
| $TiO_2$ | .48 |
| $ZrO_2$ | .06 |
| $P_2O_5$ | trace |
| $FeS_2$ | .10 |
| | 100.04 |

The large amount of water present is difficult to explain. The inclusions in the quartz account for some, and assuming the presence of some water due to incipient alteration in the sillimanite, there is still a large amount unaccounted for.

From the analysis, neglecting the water and accessory minerals, the composition of the rock is calculated as consisting of

| | |
|---|---|
| Quartz | 69 |
| Sillimanite | 31 |

Deducting the water and other accessory minerals of the rock we obtain the following comparison:

| | Analysis recalculated. | Calculated for above composition. |
|---|---|---|
| $SiO_2$ | 80.3 | 80.5 |
| $Al_2O_3$ | 19.7 | 19.5 |

As we approach the middle of the dike the rock loses its schistose structure, becomes coarser, and we find occasional masses of dumortierite, which are more and more abundant, until finally we come to what is called the lower part of the dike, consisting of abundant dumortierite and quartz. Very rarely a small black tourmaline or garnet is seen. Muscovite is more or less abundant throughout the entire lower part, and in places forms an essential part of the rock.

The lower part of the dike differs in many ways from the upper. No schistose structure is apparent; the rock is very coarse and has a mottled appearance, due to the dark bunches of dumortierite on the white quartz background.

Some masses of dumortierite are very large, examples 5 by 3 by 3 cm. being not uncommon. They usually have a radiated columnar structure and break off nearly parallel to the base in fairly plane surfaces, which on large pieces become somewhat bowl shaped. As the various fibers are not strictly parallel but radiating, the breakage can only be approximately at right angles to the general elongation of the fibers. These masses of dumortierite weather from the rock and stand out very prominently. They usually have numerous cleavage cracks parallel to the base. The color of the mineral is lavender, resembling to some extent compact rubellite. Connected with this unusual color is the unusual pleochroism that the mineral possesses, being from colorless to red purple instead of blue.

The quartz is coarse grained and has a slight greasy appearance. It sometimes occurs in large masses free from dumortierite. Magnetite is present in small grains as well as titanium oxide—rutile (?). An analysis of a sample of pure quartz was made to prove the absence of any feldspar or other silicate. The analysis follows:

| | |
|---|---|
| $SiO_2$ | 98.78 |
| $Fe_2O_3$ (total) | .45 |
| CaO | .10 |
| MgO | none |
| $TiO_2$ | .61 |
| $P_2O_5$ | trace |
| | 99.94 |

A rarer phase of the rock consists of quartz, muscovite, and dumortierite, which is here present in small pieces but a few millimeters or less thick. The mica is here an essential constituent of the rock and in places becomes abundant.

Microscopically the minerals present in the lower part of the dike are seen to be dumortierite and quartz, with muscovite and sillimanite in small quantities, together with accessory magnetite, titanite, rutile (?), apatite, and zircon, as well as a number of small undetermined inclusions. The dumortierite occurs in irregular masses with ragged outline, often resembling in shape the hornblende of a diorite.

Again, it forms fan-shaped radiating masses, sometimes of large size. Irregular broken fibrous masses also occur scattered through the slide. The quartz is allotriomorphic and is but slightly cracked. It is rather full of inclusions in places. With the exception of a little muscovite there are no secondary minerals present.

The common form for the dumortierite is shown in radiated fanlike masses that vary considerably in size. When the entire piece is larger than a quadrant, parts of the black cross (seen in spherulites) are seen when the nicols are crossed. These fan-shaped pieces are probably the results of an incomplete spherulitic growth. The most perfect one consists of but half a circle. On certain parts no radiated fibers are detectable, the (prismatic) cleavage lines being perfectly parallel. Some of the pieces become decidedly fibrous toward the ends, and the various individual fibers depart somewhat from true parallelism. In between these fibers fine-grained aggregates of muscovite can often be seen.

A number of pieces showed a (secondary?) growth of fibers, which are probably in all cases dumortierite, as with high poweis a faint pleochroism can be detected. They were at first thought to be sillimanite, but they agree in all their properties—so far as can be determined—with dumortierite. These fibers can be classed in two parts.

Those in the first part seem to have been formed as a secondary growth on the main masses of dumortierite. The fibers branch out considerably and often form a radiating fringe around an entire section of the mineral. They penetrate the quartz grains, and interstitial muscovite is absent. The line where they join the main mass of the mineral is usually fairly well defined. In general, the fibers are normal to the edge of the main mass, but locally they vary considerably, especially where they form fanlike groups.

The fibers of the second class clearly represent a stage in the alteration of the dumortierite to muscovite. The solid mass of dumortierite becomes fibrous and, at the edges, breaks up into small fibers, which gradually become loose from the parent mass. The space where they formerly joined is now occupied by a granular mass of muscovite, which also fills the spaces between the fibers. Gradually this process goes on until, finally, we have a large mass of granular muscovite in which are embedded a few fibrous prisms of the original mineral. Fig. 6 shows diagrammatically a stage in the process.

Basal sections present an entirely different appearance. The macropinacoidal cleavage, so well developed on the New York mineral, does not show on the sections of the California mineral. The imperfect prismatic cleavage is present and divides the section up into a multitude of irregular bodies. Occasionally there is a short crack parallel to the brachypinacoid, and it may be that the mineral possesses an imperfect interrupted brachypinacoidal cleavage.

The prismatic cleavage was found to be parallel to the prism {210}. A large number of measurements gave as the angle {010} ∧ cleavage cracks 62°–67°, averaging 64°–65°. The angle {010} ∧ {210} is 66° 01′. The cleavage cracks are irregular, and it was difficult to obtain any accurate measurements, but the ones made are sufficient to determine the cleavage. They were measured against the trace of the optic axial plane.

Polysynthetic twinning lamellæ were observed a number of times, and in one particular section (the same on which the prismatic cleavage was measured) were determinable to a certain extent. The alternate lamellæ extinguished together, the difference in angle of extinction of two adjacent lamellæ being about 31°. The line joining these lamellæ is parallel to the cleavage or to the prism {210}. The trace of the axial plane is parallel (or nearly so) in adjacent lamellæ.

The pleochroism of the mineral is beautiful, especially if the section be not too thin. $\mathfrak{c} = a$ is colorless, $\mathfrak{b} = b$ is colorless to very faint pink, $\mathfrak{a} = c$ is deep-red purple. None of the sections entirely possess the ordinary blue pleochroism. In some slides, however, there are certain small areas of varied and irregular shape which do show the ordinary pleochroism. These small blue areas in the rich red-purple background make a most beautiful combination. On some sections a large area will have a faint bluish-purple color, as if intermediate between the deep-blue and the red purple.

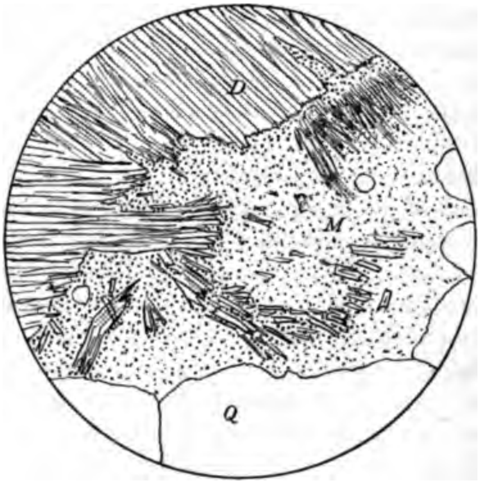


FIG. 6.—Dumortierite altering to muscovite.

This California dumortierite contains about 1½ per cent titanium oxide, which is considered as $Ti_2O_3$, replacing part of the alumina. Correlating these facts with the other known instances, it seems as if the presence of the titanium controlled the occurrence of this purple pleochroism. Samples of the Washington and Arizona mineral, free from titanite and rutile, and which possess the ordinary colorless to blue pleochroism, were tested qualitatively for titanium with hydrogen peroxide, with a negative result in each case. One might conclude, therefore, that in those areas of a dumortierite section in which the color is blue there is no titanium; that they are molecular centers, so to speak, which are free from titanium. The absorption of the mineral is $\mathfrak{a}>\mathfrak{b}=\mathfrak{c}$. Optically it is negative, the acute bisectrix $\mathfrak{a}$ being normal to the base. The axial plane is parallel to the brachypinacoid. The birefringence is higher than that of quartz, though only by a small amount.

The sections of dumortierite show that inclusions are rare. The masses of mineral are usually pure, muscovite being about the only mineral included, and this is not strictly an inclusion, as it is a secondary product, forming along the fibrous cracks of the mineral. In one basal section a number of rutile (?) crystals were observed, but such inclusions are very rare.

The quartz occurs in large allotriomorphic grains, but little cracked and sometimes almost free from inclusions and again thickly crowded with them. In no way does the quartz show the effect of pressure as the quartz of the upper part of the dike does. Occasionally when the nicols are crossed the color of a quartz grain will be somewhat banded, these bands occurring on several grains, being parallel throughout the entire slide. The inclusions are sillimanite, magnetite, rutile (?), and an undetermined mineral.

The rock composing the dike is doubtless an igneous plutonic rock. There would seem to have been an interruption in the stage of formation of the entire dike. The upper or sillimanite-bearing part was more or less completely formed when a mass was pushed into the place now occupied by the lower part of the dike. This mass exerted a great pressure on the already partly formed upper part, giving it its schistose structure. Boric acid in large amount was in the lower mass. Possibly it was a later intrusion and may have caused the increase in pressure of the lower mass, which gradually cooled and formed a borosilicate (dumortierite) and quartz. The absence of dumortierite in the upper part would seem to indicate that the addition of boric acid to the entire mass was a later incident, and had the original magma cooled without this extra intrusion the rock would have been uniform and would have consisted of an aluminum silicate (sillimanite or andalusite) and quartz. The intrusion of the boric acid could not change the character of the upper part, which had already (at least partially) formed, but it did change the character of the lower part. May it be that the pressure determined the formation of sillimanite instead of andalusite?

## WASHINGTON.

Ford mentions this occurrence of dumortierite and gives as the locality "the headwaters of the North Fork of the Washougal River in Skamania County, Wash." Mr. Brereton, of Woodstock, Oreg., very kindly sent the writer a number of specimens, which form the basis of the following notes.

As Ford states, the dumortierite occurs in a new habit, being present *as small spherulites* up to 3 mm. in diameter.

The rock in which they occur is a light-colored, fine-grained rock, containing, besides the blue spherulites, patches of a compact, soft, light-green mineral, shown to be muscovite. While the exposed surface of the rock is dark, the fresh portion light gray, rarely becoming almost white. Sometimes the dumortierite occurs so abundantly as to color the entire rock blue. Pyrite is also present and in places is very abundant.

Under the microscope the rock is seen to be very fine grained and to consist of quartz, muscovite, and andalusite in nearly equal proportions. The porphyritic spherulites of dumortierite form a striking feature in the fine-grained rock. The pale-green patches before mentioned are seen to be muscovite. Pyrite is abundant, and a few grains of magnetite (ilmenite?) surrounded by leucoxene are also present. In places the leucoxene is abundant.

The andalusite occurs as short laths with a square cross section and also as irregular more or less opaque masses without definite shape. They are probably in some stage of alteration, possibly to muscovite. While usually very minute, the laths occasionally become quite large relatively. In general, the andalusite surrounding the larger spherulites is much coarser than the main mass, though there are numerous exceptions. When clear and transparent, the laths exhibit the normal properties of andalusite. The relief is moderate and the birefringence low. Imperfect prismatic cleavage is sometimes seen. The prisms are nonpleochroic and pleochroic halos are absent. Extinction is always parallel. The elongation is always in the direction of maximum elasticity.

Sillimanite seems to be entirely absent. Muscovite occurs in the green masses and also in minute particles distributed throughout the entire rock. Nowhere does it occur in plates, as it normally does in granite, but everywhere it is in minute pieces. A good deal is probably derived from the alterations of the andalusite and dumortierite, and it may be possible that all of it is secondary—that the rock originally consisted essentially of quartz and andalusite. An analysis of the green mineral was made on a small amount of material, containing also some andalusite and quartz. It shows the mineral is muscovite, thus corroborating the microscopic evidence.

| | |
|---|---|
| $SiO_2$ | 50.13 |
| $Al_2O_3$ | 32.37 |
| $Fe_2O_3$ (total) | 1.52 |
| MgO | .09 |
| CaO | .15 |
| $H_2O-$ | 1.74 |
| $H_2O+$ | 5.08 |
| $K_2O$ | 9.60 |
| | 100.68 |
| Specific gravity | 2.80 |

The quartz occurs in allotriomorphic grains, which are so minute and so mixed with the andalusite and muscovite that recourse was had to chemical analysis to verify their determination as quartz.

The dumortierite occurs in small spherulites scattered throughout the rock and occasionally bunched together to form a large patch of blue. On an average they reach a diameter of from one-half to 1 mm. Their shape, while in general circular, is often elliptical and may become very irregular in places. They consist of fibers radially arranged and show the optical phenomena of "spherulites." The dumortierite has parallel extinction, and its birefringence is somewhat higher than that of the quartz and also slightly more than that of the andalusite, though the difference between that of the latter two minerals is very small.

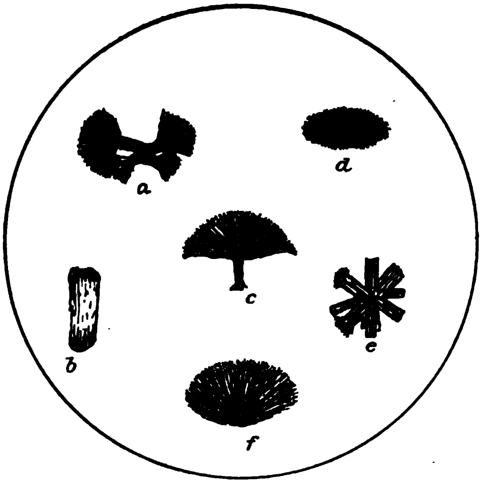


FIG. 7.—Forms of dumortierite.

The intensity of the pleochroism of the dumortierite varies, so that in some spherules there are concentric bands of fibers differing greatly in the depth of their color. Some spherules are almost colorless, and it was at first thought they might be parallel growths of andalusite with the dumortierite, but such a conclusion could not be verified.

The fibers are not always perfectly radial. They are at times gathered into "brushes," and a number of these put together may form a spherulite. The fibers are thus more thickly crowded in some places than elsewhere, and this frequently results in intensifying the pleochroism, so that in some spherulites there are numerous blotches of blue of much deeper color than the rest. Muscovite is frequently plentiful in a spherulite, being formed between the fibers, and is probably an alteration product of the dumortierite. Frequently a mass of dumortierite will be almost completely changed to mica, leaving but small fragments of the original mineral behind.

The fibers of dumortierite, while usually arranged radially, sometimes assume different shapes, and some of the masses of dumortierite seen under the microscope are reproduced in fig. 7.

Fig. 7 shows variations from typical spherulitic form that the fibers of dumortierite assume. *a* is four prisms irregularly joined at the center, the ends spreading out into "brushes" of fine fibers. This form is fairly abundant. *b* forms a prismatic portion becoming fibrous at both ends, and while common in the New York, Arizona, and California occurrences, is a rare type for Washington dumortierite. *c* forms a sort of fan and is composed of several "brushes" joined together. It is a common type. *d* is an elliptical spherulite. *e* is a mass of prisms such as *b*, but not spreading out as *a* does. It is of rarer occurrence. *f* is a bunch of radial fibers with no prismatic stem as in *c*. It is fairly common.

To confirm the microscopical determination of the minerals of the rock a chemical analysis was made by the writer of a fresh sample of the rock. The results obtained are shown below.

| | |
|---|---|
| $SiO_2$ | 57.18 |
| $Al_2O_3$ | 34.10 |
| $Fe_2O_3$ | .54 |
| FeO | .28 |
| MgO | .10 |
| CaO | .63 |
| $Na_2O$ | .39 |
| $K_2O$ | 2.57 |
| $H_2O-$ | .69 |
| $H_2O+$ | 2.02 |
| $TiO_2$ | .66 |
| $ZrO_2$ | .02 |
| $CO_2$ | none |
| $P_2O_5$ | .53 |
| $FeS_2$ | .28 |
| MnO | none |
| BaO | .04 |
| SrO | trace? |
| $Li_2O$ | none |
| | 100.03 |

Out of 200 grams of rock, 4.3 grams of dumortierite were obtained. Some was doubtless lost, and the sample still contained some andalusite, but 2 to 2½ per cent would approximately represent the amount of dumortierite present.

The mineralogical composition of the rock is about as follows:

| | |
|---|---|
| Andalusite | 35 |
| Dumortierite | 2 |
| Quartz | 32 |
| Muscovite | 27 |
| Accessories | 4 |
| | 100 |

The accessories consist of pyrite, magnetite, possibly ilmenite, leucoxene, and apatite. Considering the dumortierite as andalusite and calculating the soda in the analysis as potash, we get the following comparison:

| | Calculated composition. | Found. |
|---|---|---|
| $SiO_2$ | 57.66 | 57.18 |
| $Al_2O_3$ | 33.81 | 34.10 |
| $K_2O$ | 3.19 | 3.16 |
| $H_2O$ | 1.22 | 2.02 |
| Accessories | 4.12 | 3.54 |
| | 100.00 | 100.00 |

It is interesting to compare the minerals from the three western occurrences, where feldspar is entirely absent, and the rock in each case is composed (essentially) of the following materials:

Arizona: Dumortierite, quartz, muscovite, kyanite.
California: Dumortierite, quartz, muscovite, sillimanite.
Washington: Dumortierite, quartz, muscovite, andalusite.

As the three minerals, kyanite, sillimanite, and andalusite have the same chemical composition, the composition of the entire rock is probably also similar.

## CRYSTALLOGRAPHY.

While the writer was at the California locality he made special search for crystals of dumortierite, but is forced to the conclusion that good, perfect crystals of the mineral are yet to be found. A large quantity of seemingly promising material was collected and afterward carefully examined with a lens. One minute crystal was found showing several fairly good faces in the prismatic zone and, when mounted on the goniometer, presenting minute terminal planes giving poor reflections. Further search afforded another crystal showing several prismatic faces but no terminations.

The collection of specimens of dumortierite from Clip, Ariz., in the United States National Museum, was examined, and on one specimen was found a small crystal showing two prism faces and a dome. Further, three small crystals were obtained from the New York mineral. On this poor material the following work is based. The results obtained are perhaps not accurate, but are the best that could be obtained. The California and New York crystals are long prismatic, while the Arizona crystal may be termed short prismatic.

The mineral is orthorhombic, and from the various measurements the following axial elements have been calculated:

$$\breve{a}=.8897$$
$$\breve{c}=.6871$$

The crystals were measured with the two-circle goniometer, and the values for $p_o$ and $q_o$, from which the above ratios are obtained, are

$$p_o=.7723$$
$$q_o=.6871$$

The forms observed are as follows:

$$b=0\infty=010$$
$$a=\infty 0=100$$
$$l=\infty 2=120$$
$$m=\infty\ =110$$
$$g=\frac{3}{2}\infty=320$$
$$n=2\infty=210$$
$$d=\frac{1}{2}0\ =102$$
$$v=\frac{2}{3}0\ =203$$

Besides these forms, reflections were obtained from several other minute faces, but they were so minute, and the reflections so poor, that their validity is very questionable and they are omitted from the foregoing list. The averages of the measured angles, compared with the calculated values, are shown in the following table:

| Letter and symbol. | Measured. | | Calculated. | |
|---|---|---|---|---|
| | $\phi$. | $\rho$. | $\varphi$. | $\rho$. |
| | ° ′ | ° ′ | ° ′ | ° ′ |
| $b=0\infty=010$ | 0 00 | 90 00 | 0 00 | 90 00 |
| $a=\infty 0=100$ | 91 23 | 90 00 | 90 00 | 90 00 |
| $l=\infty 2=120$ | 29 19 | 90 00 | 29 20 | 90 00 |
| $m=\infty\ =110$ | 49 01 | 90 00 | 48 20 | 90 00 |
| $g=\frac{3}{2}\infty=320$ | 59 49 | 90 00 | 59 19 | 90 00 |
| $n=2\infty=210$ | 65 23 | 90 00 | 66 01 | 90 00 |
| $d=\frac{1}{2}0\ =102$ | 91 21 | 21 46 | 90 00 | 21 07 |
| $r=\frac{2}{3}0\ =203$ | 87 39 | 26 58 | 90 00 | 27 14 |

The brachypinacoid is present on four crystals, usually as broad faces. Some give relatively good reflections.

The macropinacoid occurs on all crystals, except the one from Arizona, and always as narrow faces. The reflections were poor, but serve to identify the form. The $\phi$ angles measured are as follows:

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| California | 90 | 02 | 90 | 00 |
| Do | 93 | 46 | | |
| Do | 90 | 00 | | |
| New York | 92 | 57 | | |
| Do | 91 | 15 | | |
| Do | 90 | 16 | | |

The brachy prism $l=\infty 2=120$ occurs on four crystals as narrow faces giving fair reflections.

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| California | 30 | 00 | 29 | 20 |
| Do | 28 | 00 | | |
| Do | 29 | 22 | | |
| New York | 29 | 34 | | |
| Do | 29 | 39 | | |

The unit prism is poorly developed on the California crystals, being present but once as a narrow face giving a poor reflection. On the Arizona crystal it occurs twice as broad faces.

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| California | 48 | 52 | 48 | 20 |
| Arizona | 49 | 16 | | |

The macro prism $g=\frac{3}{2}\infty=320$ is present several times. The measured angles vary somewhat.

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| California | 58 | 55 | 59 | 19 |
| Do | 59 | 42 | | |
| Do | 60 | 36 | | |
| Do | 59 | 38 | | |
| Do | 59 | 23 | | |
| Do | 60 | 14 | | |
| New York | 59 | 04* | | |
| Do | 60 | 07 | | |
| Do | 59 | 25 | | |
| Do | 60 | 23 | | |
| Do | 60 | 31 | | |

The macro prism $n=2\infty=210$ is present four times. The reflections were fair.

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| California | 65 | 26 | 66 | 01 |
| Do | 65 | 35 | | |
| Do | 65 | 19 | | |
| Do | 65 | 12 | | |

The macro dome $d=\frac{1}{2}0=102$ is present once on the Arizona crystal. The reflections were not very good.

The dome $v=\frac{2}{3}0=203$ is present on one of the California crystals. The face is exceedingly minute and the reflection was poor.

On crystal No. 4 (New York) were observed two very small faces, giving poor reflections.

| | $\varphi$. | | $\rho$. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| (1) | 83 | 46 | 17 | 54 |
| | 83 | 16 | 17 | 44 |
| (2) | 80 | 19 | 16 | 50 |
| | 84 | 17 | 17 | 28 |

These agree approximately with the symbol {8.1.20}, or considering the form vicinal to a dome, to {205}.

The combinations seen on the six crystals are as follows:

| | | |
|---|---|---|
| No. 1 | California | *b*, *a*, *l*, —, *m*, *g*, *n*, —, *v*. |
| No. 2 | California | *b*, *a*, *l*, *h*, —, *g*, *n*, —, —. |
| No. 3 | Arizona | —, —, —, —, *m*, —, —, *d*, —. |
| No. 4 | New York | *b*, *a*, —, —, —, *g*, —, —, —. |
| No. 5 | New York | *b*, *a*, *l*, —, —, *g*, —, —, —. |
| No. 6 | New York | —, *a*, *l*, —, —, *g*, —, —, —. |

Fig. 8 shows three of the crystals. A is from California; size, 1 mm. × ¼ mm. B is from New York; size, 1 mm. × ½ mm. C is from Arizona; size, 1 mm. × 1 mm.

The following table gives the calculation of the forms of dumortierite based on the elements given in this paper. The table corresponds to the forms given in Goldschmidt's Winkeltabellen.

| | | | | | |
|---|---|---|---|---|---|
| $a=.8897$ | $\lg a=9.94924$ | $\lg a_0=0.11202$ | $\lg p_0=9.88779$ | $a_0=1.2943$ | $p_0=.7723$ |
| $c=.6871$ | $\lg c=9.83702$ | $\lg b_0=0.16298$ | $\lg q_0=9.83702$ | $b_0=1.4554$ | $q_0=.6871$ |

| No. | Letter. | Sym. | | $\phi$. | $\rho$. | $\xi_0$. | $\eta_0$. | $\xi$. | $\eta$. | $x$ (Prism) $x:y$. | $y$. | $d=\operatorname{tg}\rho$. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Miller. | gdt. | | | | | | | | | |
| | | | | ° ′ | ° ′ | ° ′ | ° ′ | ° ′ | ° ′ | | | |
| 1 | *b* | 010 | $0\infty$ | 0 00 | 90 00 | 0 00 | 90 00 | 0 00 | 90 00 | 0 | $\infty$ | $\infty$ |
| 2 | *a* | 100 | $\infty 0$ | 90 00 | 90 00 | 90 00 | 0 00 | 90 00 | 0 00 | $\infty$ | 0 | $\infty$ |
| 3 | *l* | 120 | $\infty 2$ | 29 07 | 90 00 | 90 00 | 90 00 | 29 07 | 60 53 | .5570 | $\infty$ | $\infty$ |
| 4 | *m* | 110 | $\infty$ | 49 01 | 90 00 | 90 00 | 90 00 | 49 01 | 40 59 | 1.1510 | $\infty$ | $\infty$ |
| 5 | *g* | 320 | $\frac{3}{2}\infty$ | 59 47 | 90 00 | 90 00 | 90 00 | 59 47 | 30 13 | 1.7170 | $\infty$ | $\infty$ |
| 6 | *n* | 210 | $2\infty$ | 65 23 | 90 00 | 90 00 | 90 00 | 65 23 | 24 37 | 2.1825 | $\infty$ | $\infty$ |
| 7 | *d* | 102 | $\frac{1}{2}0$ | 90 00 | 21 07 | 21 07 | 0 00 | 21 07 | 0 00 | .3862 | 0 | 0.3862 |
| 8 | *v* | 203 | $\frac{2}{3}0$ | 90 00 | 27 14 | 27 14 | 0 00 | 27 11 | 0 00 | .5147 | 0 | .5147 |

The minerals andalusite, sillimanite, staurolite, and dumortierite possess certain properties that are very much alike. They are all orthorhombic, their axial ratios are similar, their principal constituents are silica and alumina, and in many ways these minerals can well be grouped together. Using the word isomorphous in its general loose sense, these minerals are isomorphous. A comparison of their axial ratios is given below:

| | ă. | ċ. |
|---|---|---|
| Andalusite | .9861 | .7025 |
| Sillimanite | .970 | (?) |
| Staurolite | .9795 | .6942 |
| Dumortierite | .8897 | .6871 |

For staurolite the values are taken from Goldschmidt's Winkeltabellen with the *a* and *c* axes interchanged.

There was some question in the writer's mind as to whether the orientation chosen was the best or whether the present prism {210} had not better be made the unit prism. The cleavage is parallel to this prism, and the twinning also bears some relation to this form. Moreover, the ratio above given for staurolite, though adopted by Goldschmidt, is not, in the writer's opinion, the best one for the mineral. The one given by Dana, who makes the *a* axis just half as long, would be the better one. The common form of staurolite is prismatic, with a prism angle of 50° 40′ (Dana).

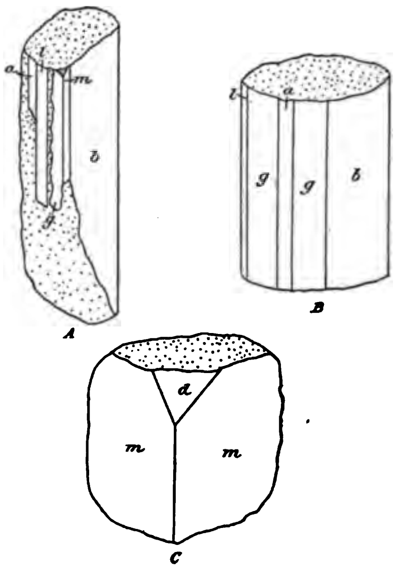


FIG. 8.—Dumortierite.

For andalusite, on the other hand, the prismatic angle is 89° 12′ and the *a* axis should be .9861, as given. Should dumortierite be classed crystallographically with staurolite or with andalusite? Unfortunately, the evidence is almost too meager to decide this question. The unit prism is poorly developed on the California crystals and entirely absent from the New York, the strongest form next to the brachypinacoid being the prism {320}. The prismatic cleavage is parallel to {210}, and making the cleavage form the unit prism the *a* axis should be given half its present value. On the other hand, the Arizona crystal is of the typical andalusite habit, and, making the prism the unit one, we obtain the axes as here given. Giving staurolite the (approximate) same axes as andalusite and dumortierite, the prismatic cleavage is parallel to the same form as in dumortierite {210}.

It may be worth noting that as staurolite and dumortierite are related crystallographically more closely than either with andalusite, the chemical composition of the two former is much more complex than that of the last named species.

## PHYSICAL PROPERTIES.

### GENERAL PROPERTIES.

Macroscopic crystals are exceedingly rare, the mineral usually occurring in prismatic fibrous forms showing no crystal faces. The Washington mineral occurs in spherulitic forms which are but a special arrangement of fibers.

The cleavage is—macropinacoidal good, prismatic {210} imperfect, and basal imperfect, and perhaps, also, in traces, brachypinacoidal. The New York mineral shows the macropinacoidal cleavage very well, while the California mineral shows the imperfect basal cleavage. The fracture is rough and the fibrous prisms are somewhat brittle, though a mass of fibers is very tough. The hardness is 7.

The following values for the specific gravity have been given:

3.36
3.265
3.22
3.319
3.226-3.43
3.211-3.302

Av. 3.292

The writer obtained 3.306 as the density of a piece weighing about 10 grams. A good average value would be 3.30.

The luster is vitreous, somewhat satiny on the fibrous California mineral. The color is usually blue, though also lavender and greenish blue to black. The colors noted are:

France: Blue, black.
Germany: Blue, bluish green.
Sweden: Blue.
New York, Arizona, Washington: Blue.
California: Lavender.

The mineral is usually opaque in its massive form, but the small fibers of the California mineral are transparent to translucent.

## OPTICAL CHARACTERS.

All the determinations made on the mineral from different localities have given the same orientation. The axial plane is parallel to the brachypinacoid, the acute bisectrix is normal to the base. The mineral is negative.

The following are the pleochroisms noted on the mineral from different localities: Colorless to cobalt blue, pale yellow or colorless to cobalt blue, colorless to pistachio green, colorless to rose salmon, colorless to deep carmine, colorless to red purple (California).

### AXIAL ANGLES.

On the Norway mineral, $2V_{Na}$ was determined as about 35°. Linck states that the angle is somewhat less than that of muscovite.

The axial angle on the California dumortierite is small. The following measurements were made with the microscope and micrometer ocular, and represent only approximate values:

$$2E_{Li}=33^\circ$$
$$2E_{Na}=37$$
$$2E_{Cu}=42$$

The dispersion thus is $\rho<v$.

### INDICES OF REFRACTION.

Linck obtained for the dumortierite from Germany

$$\alpha=1.678$$
$$\beta=1.686$$
$$\gamma=1.689$$
$$\alpha-\gamma=.011$$

Michel Levy and Lacroix give $(\gamma-\alpha)=.010$ with the dispersion (strong) $\rho>v$, while Bertrand in his original description gives $\rho<v$ as the dispersion. The value $(\gamma-\alpha)$ for quartz is .009 and for andalusite .011. The birefringence for dumortierite is always a little higher than that of the quartz, and in the Washington mineral very nearly that of the andalusite.

### PYROGNOSTICS.

The mineral is infusible before the blowpipe, but loses its color, becoming white. It gives a blue color on heating with cobalt nitrate. If the mineral be very finely powdered and intimately mixed with potassium bisulphate and calcium fluoride and the powder be carefully introduced into the flame, a green color, due to the boric acid, may be momentarily seen, but it is an exceedingly difficult reaction to obtain.

## CHEMICAL COMPOSITION.

In 1881, after the first announcement of the discovery of the mineral had been made by Gonnard, Damour gave the following analysis of the mineral:

| | |
|---|---|
| $SiO_2$ | 29.85 |
| $Al_2O_3$ | 66.02 |
| $Fe_2O_3$ | 1.01 |
| MgO | .45 |
| Ign | 2.25 |
| | 99.58 |
| Specific gravity | 3.36 |

From this analysis the formula $4Al_2O_3,3SiO_2$ was calculated. Damour did not suspect the presence of boric acid in the mineral, and what was weighed as alumina probably contained several per cent

$B_2O_3$. In 1887 Riggs gave an analysis of the New York mineral, in which he found 4 per cent $B_2O_3$. Two years later Whitfield gave several analyses of American dumortierite, all showing the presence of $B_2O_3$. In 1899 Linck gave some observations on the mineral, including an analysis by W. Schimpff, showing a strong test for boric acid. In 1902 Ford gave us three analyses of American dumortierite and also mentioned two new localities. During the winter of 1903–4 the writer made an analysis of the California dumortierite collected by himself during the previous summer. An analysis was also made at the same time of the Washington dumortierite kindly furnished by Mr. Brereton.

Believing that one good analysis is better than several poorer ones for the exact determination of the formula of the mineral, there will be presented an analysis of the California dumortierite made by the writer. From this a formula has been calculated for the mineral, and it will be shown how the other analyses agree with this proposed formula. Before giving the results a few preliminary words will not be out of place.

An analysis of dumortierite is a difficult operation. The small amount of silica makes it difficult to get a good fusion. On preliminary trials on an analysis of the California dumortierite, it was found that what was weighed as silica in the regular course of the analysis contained more or less of the undecomposed mineral. Results of about 30 to 32 per cent were obtained. This is, of course, on the uncorrected silica. It was found that a second fusion of the supposed silica was not only very beneficial, but also necessary. What was then weighed as silica contained but little residue. The handling of so large an amount of alumina is very cumbersome, and an accurate determination of the alumina (plus iron and other oxides here precipitated) is difficult. The boric acid determination is tedious and difficult. The accurate determination of the water, which is given off only at a high temperature, is also not easy. One can thus see that an analysis of such a mineral is a troublesome operation and that a greater allowance must be made than for most silicate analyses.

The specimen analyzed was selected in the field, an exceedingly pure piece weighing about 10 grams being chosen. The specific gravity of this piece was taken by suspension in water, giving 3.306. This was broken into small pieces and carefully examined for muscovite, quartz, or other minerals. Sections of the mineral showed that the dumortierite was free from any inclusions. No grains of any titanium mineral abundant in the quartz could be detected. When the mineral was powdered it was treated with heavy solution, of specific gravity 3.10, and a minute amount of mineral (muscovite?) stayed on top and was removed. The separation was repeated several times, and the powder was dried at 100° and carefully examined under the microscope for impurities. None was found. The mineral was unquestionably pure.

The general analytical methods used were those pursued in the Survey laboratory for silicate analyses, with a few modifications. The mineral was fused with sodium carbonate, the silica separated and re-fused with sodium carbonate. The silica was filtered off, weighed, and treated with HF, which left some residue. The alumina (plus iron and titanium) was precipitated three times to be sure of removing all of the sodium salts. It was ignited and weighed with the silica residue, fused with sodium bisulphate, some silica recovered, the iron was reduced and determined by titration and the titanium was determined colorimetrically. The presence of titanium was suspected from the color of the pleochroism of the mineral. The boric acid was determined by the Gooch method, using all of the known precautions. The mineral was twice fused with sodium carbonate and the boric acid was finally weighed as calcium borate. The water was collected in a calcium chloride tube, the mineral being heated in a Gooch tubulated crucible in the usual manner. A blank determination was run before and after each water determination and a small correction applied. All possible precautions were taken throughout the analysis, which was made in duplicate. The results are given below.

| | 1. | 2. | Average. |
|---|---|---|---|
| $SiO_2$ | 28.58 | 28.78 | 28.68 |
| $Al_2O_3$ | 63.31 | 63.30 | 63.31 |
| $Ti_2O_3$ | 1.49 | 1.40 | 1.45 |
| $Fe_2O_3$ | .21 | .25 | .23 |
| $H_2O$ | 1.53 | 1.51 | 1.52 |
| $B_2O_3$ | 5.21 | 5.53 | 5.37 |
| | 100.33 | 100.77 | 100.56 |

The titanium is regarded as present as $Ti_2O_3$, replacing the alumina. Combining the alumina, titanium, and iron, the following ratios are obtained:

| | | |
|---|---|---|
| $SiO_2$ | 5.94 | or 6 |
| $Al_2O_3$ | 8.00 | 8 |
| $B_2O_3$ | 1.06 | 1 |
| $H_2O$ | .96 | 1 |

The formula for dumortierite then is

$$8Al_2O_3,\ 1B_2O_3.\ 1H_2O,\ 6SiO_2.$$

It has not been proved that either boric acid and alumina or boric acid and hydroxyl may mutually replace each other in minerals as fluorine and hydroxyl are known to do. There is, then, no reason that the writer can see why the alumina, boric acid, and water in dumortierite should not be present in fixed quantities, and that the variations shown in analyses are not due to "isomorphous replacements," but to inaccuracy of analyses or impure material.

Having established the above formula from the analysis, let us see how closely the other analyses conform to it.

The best series of analyses is that made by Ford of mineral from three American localities. His first analyses of the Arizona material are tabulated below.

| | 1. | 2. | 3. | Average. |
|---|---|---|---|---|
| $SiO_2$ | 30.00 | 29.66 | 29.91 | 29.86 |
| $Al_2O_3$ | 63.20 | 63.74 | 63.76 | 63.56 |
| $Fe_2O$ | .23 | .23 | ........ | .23 |
| $B_2O_3$ | 5.47 | 5.06 | ........ | 5.26 |
| $H_2O$ | 1.45 | 1.38 | ........ | 1.41 |
| | ........ | ........ | ........ | 100.32 |

The ratios calculated from the average analysis give

| | | | |
|---|---|---|---|
| $SiO_2$ | 6.29 | or | 6×1.05 |
| $Al_2O_3$ | 7.94 | | 8× .99 |
| $B_2O_3$ | .96 | | 1× .96 |
| $H_2O$ | 1.00 | | 1×1.00 |

The agreement with the proposed formula is perfect.

His second analysis, that of the California mineral, shows slight variations from the results obtained by the writer. The presence of titanium was not determined. His analysis is:

| | | Ratios. | |
|---|---|---|---|
| $SiO_2$ | 30.58 | 6.17 or | 6×1.03 |
| $Al_2O_3$ | 61.83 | 7.40 | 8× .93 |
| $Fe_2O_3$ | .36 | | |
| $B_2O_3$ | 5.93 | 1.04 | 1×1.04 |
| $H_2O$ | 2.14 | 1.45 | 1×1.45 |
| | 100.84 | | |

The ratios agree well with the new formula except for the water content. Remembering, however, that the amount of water present is very small and that a difference of 0.10 per cent. would make a large difference in the ratios; that Ford determined his water by igniting the mineral with lime, thus not weighing the water directly; and that the writer obtained only 1.52 per cent on the same mineral from the same locality, it seems much preferable to regard the determination of the water in Ford's analysis as too high rather than to attempt an explanation of it by the assumption of "isomorphous replacement, etc."

Of the third analysis by Ford of the New York mineral, he says: "It is fully realized that the results are not to be looked upon as being as exact as those of the other analyses."

His analysis is:

| | | Ratios. | |
|---|---|---|---|
| $SiO_2$ | 31.24 | 6.23 | or $6 \times 1.04$ |
| $Al_2O_3$ | 61.26 | 7.23 | $8 \times .90$ |
| $Fe_2O_3$ | .10 | | |
| $B_2O_3$ | 6.14 | 1.06 | $1 \times 1.06$ |
| $H_2O$ | 2.09 | 1.40 | $1 \times 1.40$ |
| | 100.83 | | |

Assuming that his water content is somewhat high, the ratios agree well with the new formula.

This concludes the list of analyses of dumortierite which were made on pure material and with due knowledge of what was to be determined. A number of other analyses will now be given, which serve in a general way for the determination of the composition of the mineral, but can not be relied on for exact results.

One, which has heretofore not been published, was made by the writer on the Washington dumortierite. About 200 grams of the rock was powdered, and by repeated separation with heavy solutions about 4 grams of dumortierite were obtained. The sample was by no means pure. It was found impossible to remove all the andalusite, an unknown but small amount remaining. It was found during the course of the analysis that the samples also contained some titanite (leucoxene) and a very small amount of pyrite. The analysis was made with all the care possible, and the following results were obtained:

| | | Ratios after deducting titanite. | |
|---|---|---|---|
| $SiO_2$ | 28.51 | 5.97 | or $6 \times 0.99$ |
| $Al_2O_3$ | 59.75 | 7.80 | $8 \times .97$ |
| $Fe_2O_3$ | 2.48 | | |
| $TiO_2$ | .95 | | |
| $H_2O$ | 2.12 | 1.53 | $1 \times 1.53$ |
| $B_2O_3$ | 5.54 | 1.03 | $1 \times 1.03$ |
| CaO | .68 | | |
| | 100.03 | | |

Excepting for the high-water content the analysis agrees very well with the formula. No allowance was made for the small amount of andalusite present, which would slightly alter the ratios.

The analysis by Schimpff of dumortierite from Germany is given below. Obviously it is worthless for any calculation.

| | |
|---|---|
| $SiO_2$ | 36.81 |
| $Al_2O_3$ | 57.27 |
| CaO | 1.66 |
| MgO | 1.38 |
| Ign | 1.31 |
| $B_2O_3$ | strong test |
| | 98.43 |

Of similar character is Jannasch's analysis of the Argentine mineral. The sample was probably very impure.

| | |
|---|---|
| $SiO_2$ | 35.01 |
| $B_2O_3$ | 2.51 |
| $TiO_2$ ($Ti_2O_3$?) | 1.08 |
| $Al_2O_3$ | 51.49 |
| $Fe_2O_3$ | 1.04 |
| CaO | .28 |
| MgO | .54 |
| $K_2O$ | 3.62 |
| $Na_2O$ | .96 |
| $H_2O$ | 3.02 |
| | 99.55 |
| Specific gravity | 3.255 |

There now remain the analyses of Whitfield and Riggs, of which but one is suitable for any calculation. The analyses are as follows, No. 1 being of the New York mineral, by Riggs; No. 2 of the New York, and Nos. 3 and 4 of the Arizona mineral, by Whitfield.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $SiO_2$ | 34.82 | 31.44 | 31.52 | 27.99 |
| $Al_2O_3$ | 55.30 | 68.91 | 63.66 | 64.49 |
| MgO | .57 | ........ | .52 | trace |
| $K_2O$ | 1.04 | ........ | .11 | ........ |
| $Na_2O$ | 1.76 | ........ | .37 | ........ |
| $P_2O_5$ | ........ | ........ | ........ | .20 |
| $B_2O_3$ | 4.07 | trace | 2.62 | 4.95 |
| Ign | 2.96 | ........ | 1.34 | 1.72 |
| | 100.52 | 100.33 | 100.14 | 99.35 |

The ratios of the last analyses are

| | | |
|---|---|---|
| $SiO_2$ | 5.94 or | $6\times0.99$ |
| $Al_2O_3$ | 8.09 | $8\times1.01$ |
| $B_2O_3$ | .80 | $1\times.80$ |
| $H_2O$ | 1.22 | $1\times1.22$ |

Approximately they agree with the formula.

The only question of which there seems to be any doubt is whether the water should be 1 or 1½ molecules. As, however, the best analyses show but 1 molecule and their evidence is considered as of much more value than a larger number of inferior analyses, there is no question in the writer's mind but that the formula for dumortierite is

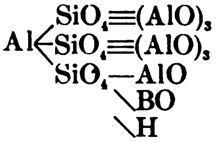
$$8Al_2O_3,\ 1B_2O_3,\ 1H_2O,\ 6SiO_2.$$

This may be written

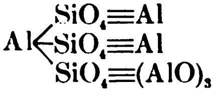
$$(SiO_4)_3Al(AlO)_7(BO)H,$$

which is the same as the formula given by Groth except that for (BO) he puts (AlO). His formula is $(SiO_4)_3Al(AlO)_8H$.

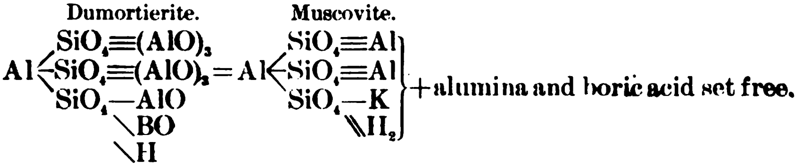
This formula may be written in the following form:



This formula is similar to that of andalusite, which is written



Dumortierite alters to muscovite. The change can be very well shown by these formulæ, and seems to be entirely in accordance with the group of minerals to which it is related.



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# MINERALOGICAL NOTES.

By W. T. Schaller.

## 1. HALLOYSITE.

The pink clay occurring at Branchville, Conn., has been shown[a] to be montmorillonite, while that occurring at Norway, Me., has proved[b] to be cimolite. That from the lepidolite mine near Pala, San Diego County, Cal., differs from both of the New England clays, being comparable with halloysite.

At Pala the halloysite occurs in large seams, often several inches thick, and many feet in length. The clay is somewhat moist, but quickly dries to a crumbling mass when taken out of the mine. In color it is rather deeper pink than the Norway cimolite and occasionally is somewhat translucent. It readily crumbles to a fine powder when placed in water. The material analyzed had been drying in the air for over three months.

The results of analysis are shown below.

| | |
|---|---|
| $SiO_2$ | 43.62 |
| $Al_2O_3$ | 35.55 |
| $Fe_2O_3$ | .21 |
| MnO | .26 |
| CaO | 1.02 |
| MgO | .19 |
| $Li_2O$ | .23 |
| $Na_2O$ | .19 |
| $K_2O$ | .03 |
| $H_2O$ (107°) | 6.63 |
| $H_2O$ (above 107°) | 12.25 |
| $TiO_2$ | none |
| | 100.18 |

The iron was determined as $Fe_2O_3$, FeO not being tested for. The analysis agrees well with the formula

$$H_4Al_2Si_2O_9+H_2O.$$

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[a] Am. Jour. Sci., vol. 20, 1880, p. 283.

[b] Ibid., 32, 1886, p. 355.

## 2. AMBLYGONITE.

The occurrence of amblygonite at the lepidolite mine at Pala, Cal., has already been noticed, and a somewhat fuller description of the mineral is here given. A large deposit has been uncovered, and the indications seem to show that it is merely a small part of an extensive body of massive amblygonite. The mineral usually occurs pure; very rarely small amounts of lepidolite are present with it. Frequently broad cleavage faces with irregular outlines can be seen. The color is white and in thin pieces the mineral is translucent. It fuses easily, coloring the flame red, and in powder is difficultly decomposed by sulphuric acid.

An analysis of a specimen kindly presented by Mr. G. F. Kunz gave the writer the following results:

| | |
|---|---|
| $P_2O_5$ | 48.83 |
| $Al_2O_3$ | 33.70 |
| $Fe_2O_3$ | .12 |
| MnO | .09 |
| MgO | .31 |
| $Li_2O$ | 9.88 |
| $Na_2O$ | .14 |
| $H_2O$ | 5.95 |
| F | 2.29 |
| $TiO_2$ | none |
| | 101.31 |
| Less O | .96 |
| | 100.35 |

Regarding fluorine and hydroxyl as isomorphous, in the specimen analyzed the latter greatly predominates over the fluorine. The water was determined by igniting the mineral with lead oxide, previously heated nearly to fusion. Three determinations gave the loss of weight, due to the escape of water, as 5.89, 6.01, 5.95 per cent. The loss of the mineral on ignition was 8.03 per cent, equaling the sum of the water and fluorine content.

## 3. BOOTHITE.

A specimen of a pale-blue copper sulphate was collected at the copper mine near Campo Seco, Calaveras County, Cal., by Mr. James Wise and kindly presented to the writer for investigation. The pale-blue color suggested that the mineral might be boothite instead of the more frequently occurring chalcanthite. The results of a chemical analysis have shown that the mineral is boothite, thus affording a second locality for this interesting mineral. Careful quantitative determinations of hydrous copper sulphates will probably show that the heptahydrate is not so rare as may be supposed.

The mineral from Campo Seco occurs massive, showing no crystalline structure. The average of several determinations afforded the results shown below.

| | | Ratio. |
|---|---|---|
| CuO | 26.13 | 1.00 |
| FeO | .81 | .03 |
| MgO | .64 | .05 |
| $SO_3$ | 27.25 | 1.04 |
| $H_2O$ (110°) | 36.76 } 41.67 | 6.22 } 7.06 |
| $H_2O$ (above 110°) | 4.91 | .84 |
| Insol | 3.96 | ............ |
| | 100.46 | ............ |

Formula, $CuSO_4.H_2O+6H_2O$.

A careful determination of the specific gravity of the mineral gave (21° C.) 1.944. This being much lower than the value obtained (2.1) on the boothite from Leona Heights, a redetermination of the latter was made on purer material collected since the publication of the first results,[a] This gave as an average value the figure (22° C.) 1.935. The average of these two determinations, or 1.94, is probably very near the true value for the specific gravity of boothite.

## 4. PISANITE.

A small specimen of massive pisanite from Gonzales, Monterey County, Cal., was analyzed some time ago and the results are here presented. The quantity of material available for analysis was very small and the determinations do not claim any great accuracy. About 6 per cent of insoluble matter has been deducted and the results recalculated to 100 per cent.

| | | Ratio. |
|---|---|---|
| CuO | 7.56 | 0.27 } 0.89 |
| FeO | 15.85 | .72 |
| $SO_3$ | 30.74 | 1.08 |
| $H_2O$ | 45.85 | 7.18 |
| | 100.00 | ............ |

The analysis approximates to the formula $CuO.2FeO.3SO_3.21H_2O$.

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[a] Schaller, W. T., Minerals from Leona Heights, Alameda County, Cal.: Bull. Dept. Geol. Univ. Cal., vol. 3, No. 7.

In the following table all available analyses of pisanite are tabulated, and one can readily see that there is no definite ratio between the copper and iron. The formula is then written $(Cu,Fe)SO_4.7H_2O$, pisanite being an isomorphous mixture of melanterite and boothite. All the analyses are calculated to 100 per cent.[a]

| | No. 1. | No. 2. | No. 3. | No. 4. | No. 5. | No. 6. |
|---|---|---|---|---|---|---|
| CuO | ........ | 7.56 | 9.17 | 10.07 | 12.61 | 15.52 |
| FeO | 25.86 | 15.85 | 16.37 | (16.15) | 14.14 | 12.14 |
| $SO_3$ | 28.80 | 30.74 | 29.00 | 28.84 | 28.44 | 27.82 |
| $H_2O$ | 45.34 | 45.85 | 45.46 | (44.94) | 44.81 | 44.52 |
| | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 |

| | No. 7. | No. 8. | No. 9. | No. 10. | No. 11. |
|---|---|---|---|---|---|
| CuO | 15.56 | 17.45 | 17.64 | 18.81 | 27.85 |
| FeO | 10.98 | 10.18 | 9.62 | 8.51 | ........ |
| $SO_3$ | 29.90 | 28.43 | 28.27 | 27.93 | 28.02 |
| $H_2O$ | 43.56 | 43.94 | 44.47 | 44.75 | 44.13 |
| | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 |

## 5. GYROLITE.

In the same rock and but about 50 feet distant from the place where the pectolite[b] and datolite occur, at Fort Point, San Francisco, Cal., were a number of veins of a pearly massive zeolite, which analysis showed to be gyrolite. The veins, ranging from microscopic size to a thickness of nearly a centimeter, occur in a dark basic igneous rock, probably a basalt. The gyrolite has replaced the rock and grown outward from both sides of the vein, forming a contact line in the middle where the two parts join. The mineral is grouped in spherical and also in massive platy or plumose aggregates. It is at times associated with apophyllite, but usually occurs pure. Its color on the fresh fracture is white, while on the weathered surface it is a brownish-yellow. It has a decided pearly luster and possesses a good cleavage.

Under the microscope, a thin section of the rock containing these veins shows that the rock has been replaced by this secondary mineral, always commencing as a minute vein, probably originating in a microscopic fissure of the rock. A multitude of these veins are present, lying in nearly parallel position and varying greatly in size. There is nothing to suggest that the gyrolite may have been derived from apophyllite. It may have been derived entirely from the igneous rock or only in part. The mineral was first deposited at the walls of the vein and from thence grew outward into the open fissure from both sides, also probably replacing the rock wall. The thin section, as well as the hand specimen, shows a decided plumose structure of the mineral. These plumose aggregates show, under crossed nicols, the black extinction cross. The interference colors are bright and vivid. Heddle[a] says of these: "The sections of orbicular gyrolite are among the most splendid of polariscopic objects, resembling a circularly opened fan, of most brilliant and abruptly contrasted colors."

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[a] Anal. No. 1, theoretical comp. of melanterite, $FeSO_4.7H_2O$. No. 2, Schaller, anal. quoted above; No. 3, Schaller, Bull. Dept. Geol. Univ. of Cal., vol. 3, No. 7; No. 4, Hintz, Zeit. Krys., 2, 309; No. 5, Hillebrand, Bull. U. S. Geol. Survey No. 220, p. 105; No. 6, see No. 3; No. 7, Pisani, Comptes Rendus, 1859, 48, 807; No. 8, see No 3; Nos. 9 and 10, Herz. Zeit. Krys., 26, 16; No. 11, theoretical comp. of boothite, $CuSO_4.7H_2O$.

[b] Eakle, A. S., Mineralogical notes: Bull. Dept. Geol. Univ. Cal., vol. 2, No. 10, 1901.

The properties of the mineral are not very well known. Heddle[b] mentions some crystals resembling heulandite in form. The mineral is described as unaxial, though as a matter of fact it is biaxial, but like biotite the axial angle becomes 0° at times. Cleavage plates show the emergence of a bisectrix. The (approximate) measurement of $2\ E_{Na}$ gave values (on the Fort Point material) of from 0° to 25°. The sign is always negative. Heddle obtained $2\ E = 0^\circ - 3^\circ$. When there was a straight edge suggesting a possible crystal direction, the extinction was not parallel.

Dana[c] gives as the formula of the mineral $3H_2O, 2CaO, 3SiO_2$, while Clarke[d] gives $5H_2O, 4CaO, 6SiO_2$, which is the formula deduced by the writer from the following analysis of the San Francisco mineral:

| | | Ratio. |
|---|---|---|
| $SiO_2$ | 53.47 | 6.02 |
| $Al_2O_3$ | .22 | .......... |
| $CaO$ | 32.00 | 4.02 |
| $Na_2O$ | 1.25 | |
| $H_2O$ | 13.21 | 4.99 |
| | 100.15 | .......... |

The specific gravity is 2.39. Traces of iron, magnesium, and potassium are present, while strontium, chlorine, and fluorine are absent.

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[a] Heddle, M. F., On the optic properties of gyrolite: Min. Mag., vol. 9, 1890–1891, p. 391.

[b] Heddle, M. F., On the crystalline form of gyrolite: Min. Mag., vol. 8, 1888–1889, p. 272.

[c] System of Mineralogy, 6th edition, p. 566, 1892.

[d] Clarke, F. W., The constitution of the silicates: Bull. U. S. Geol. Survey No. 125, 1895, p. 81.

The mineral is easily decomposed by hydrochloric acid. Certain analogies can be noted between heulandite and gyrolite, and from this and the properties of the mineral, as far as known, the writer believes gyrolite to be monoclinic. Heddle says in connection with some plane angles he measured on gyrolite from Scotland: "The above, as far as it goes, seems to show gyrolite to be oblique, and near heulandite in form."

The formulæ of the two minerals are also similar.

| | | | |
|---|---|---|---|
| Gyrolite | $6SiO_2$. $4CaO$. | | $5H_2O$. |
| Heulandite | $6SiO_2$. $Al_2O_3$. | $CaO$. | $5H_2O$. |

## 6. APOPHYLLITE.

With the gyrolite were found a number of specimens of clear, transparent, but very minute crystals of apophyllite. These are usually less than 1 mm. thick and show only the base, prism of the second order, and the unit pyramid of the first order. Some years ago a specimen was obtained here on which the apophyllite crystals have been changed to quartz. As they proved to be infusible, a more extended investigation of them was made.

The crystals, from 1 to 3 mm. in diameter, appear cubic, with the corners truncated by small faces. Measurements of the best crystals showed that they are tetragonal and that they agree in angles with those of apophyllite. The prism faces are vertically striated and faces of the other two forms are also rarely somewhat striated.

By trial it was found that hydrochloric acid does not attack the crystals, so that the associated calcite was easily removed and pure material for analysis was obtained. The crystals are opaque, and no optical determinations could be made.

The analysis gave the following figures, showing that the crystals are now, chemically, essentially silica with various impurities:

| | |
|---|---|
| $SiO_2$ | 90.58 |
| $Al_2O_3$ | 1.58 |
| $CaO$ | 1.87 |
| $MgO$ | 2.20 |
| $H_2O$ | 4.32 |
| | 100.55 |

## 7. TELLURITE.

Several small crystals of tellurite were referred to the writer by Doctor Hillebrand. Their locality is the W. P. H. mine, Cripple Creek, Colo. The crystals are transparent yellow and reach a length of about 2 mm. They are thin tabular parallel to the brachypinacoid. This form is normal to a bisectrix, and the axial plane is parallel to *a* (100).

The crystals show numerous vicinal faces, which are here not given. The forms which were established are[a]

$$b=\{010\},\ r=\{120\},\ {}^{*}h=\{520\},\ {}^{*}l=\{210\},\ p=\{111\}.$$

The angles measured are:

| | Measured. | | | | Calculated. | | | |
|---|---|---|---|---|---|---|---|---|
| | φ. | | ρ. | | φ. | | ρ. | |
| | ° | ' | ° | ' | ° | ' | ° | ' |
| b=010 | 0 | 00 | 90 | 00 | 0 | 00 | 90 | 00 |
| r=120 | 47 | 34 | 90 | 00 | 47 | 30 | 90 | 00 |
| h=520 | 79 | 30 | 90 | 00 | 79 | 37 | 90 | 00 |
| l=210 | 77 | 17 | 90 | 00 | 77 | 06 | 90 | 00 |
| p=111 | 65 | 47 | 48 | 14 | 65 | 23 | 48 | 16 |

The two new forms, *h* and *l*, gave the following measurements:

| h = {520}. | Measured, φ. | | Calculated, φ. | |
|---|---|---|---|---|
| | ° | ' | ° | ' |
| Crystal No. 1 | 79 | 01 | 79 | 37 |
| Do | 79 | 19 | | |
| Crystal No. 2 | 79 | 14 | | |
| Crystal No. 4 | 80 | 59 | | |
| Do | 79 | 02 | | |
| Do | 79 | 26 | | |

On crystals Nos. 1 and 4 the faces are very narrow, while on crystal No. 2 they are much broader. The reflections were fairly good.

| l = {210}. | Measured, φ. | | Calculated, φ. | |
|---|---|---|---|---|
| | ° | ' | ° | ' |
| Crystal No. 3 | 77 | 21 | 77 | 06 |
| Do | 77 | 12 | | |

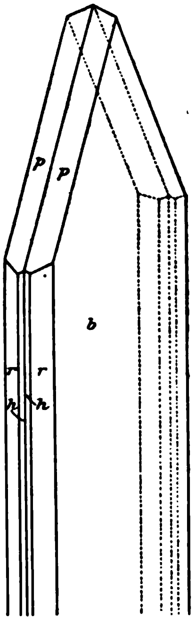

FIG. 9.—Tellurite.

The faces were both narrow and the reflections were rather poor.

It seems to the writer that the axial ratio as given by Dana should be preferred to that of Goldschmidt, who doubles both the *a* and *c* axes. This would make the (only) terminal form the unit pyramid, while the forms in the prism zone have as simple indices in one case as in another, as can be seen from the following table.

---

[a] Dana's letters and indices.

| | Dana. | Goldschmidt. | | Dana. | Goldschmidt. |
|---|---|---|---|---|---|
| *b* | 010 | 010 | *h* | 520 | 510 |
| *m* | 110 | 210 | *l* | 210 | 410 |
| *r* | 120 | 110 | *p* | 111 | 212 |
| *s* | 140 | 120 | | | |

## 8. PREHNITE.

A mineral from Lower California, sent by G. F. Kunz for determination, proved to be prehnite. The analysis gave the following results:

| | |
|---|---|
| $SiO_2$ | 43.48 |
| $Al_2O_3$ | 24.52 |
| $Fe_2O_3$ | .34 |
| CaO | 27.19 |
| $H_2O$ | 4.49 |
| | 100.02 |
| Specific gravity | 2.90 |

## 9. HORNBLENDE.

Coating the walls and projecting into the amygdaloidal cavities of a lava from Mono County, Cal., were noted small crystals of hornblende, tridymite, and biotite. The locality of the rock, which was kindly furnished by Mr. Maynard Bixby, of Salt Lake City, is given as 8 miles west of Bridgeport, Mono County, Cal. The hornblende and tridymite occur in sharp crystals, of which a few were measured.

Biotite occurs abundantly as small hexagonal plates, which, under the microscope, show at times a zonal banding in color, the darker being on the outside. The axial angle is very small—at times uniaxial.

The hornblende occurs as long prismatic crystals, becoming extremely acicular and hairlike at times. It is often grouped in radiating bunches and again occurs in single prisms. Their maximum thickness is about one-half millimeter, and in length they reach not over 5 mm. Under the microscope the pleochroism is in shades of brown. The maximum extinction was 15°.

Two terminated crystals of measurable size showed the following forms: $c=\{001\}$, $b=\{010\}$, $a=\{100\}$, $m=\{110\}$, $t=\{\bar{2}01\}$, $z=\{021\}$, $i=\{\bar{1}31\}$.

| | Measured. | | | | Calculated. | | | |
|---|---|---|---|---|---|---|---|---|
| | ϕ. | | ρ. | | ϕ. | | ρ. | |
| | ° | ′ | ° | ′ | ° | ′ | ° | ′ |
| $c=001$ | 90 | 25 | 14 | 57 | 90 | 00 | 14 | 58 |
| $b=010$ | 0 | 00 | 90 | 00 | 0 | 00 | 90 | 00 |
| $a=100$ | 90 | 14 | 90 | 00 | 90 | 00 | 90 | 00 |
| $m=110$ | 62 | 08 | 90 | 00 | 62 | 05 | 90 | 00 |
| $t=\bar{2}01$ | 90 | 03 | 40 | 36 | 90 | 00 | 40 | 05 |
| $z=021$ | 24 | 32 | 32 | 50 | 24 | 28 | 32 | 50 |
| $i=\bar{1}31$ | 18 | 18 | 42 | 45 | 18 | 03 | 42 | 49 |

Crystal No. 1 shows: *a*, *b*, *c*, *m*, *t*, *z*, *i*.
Crystal No. 2 shows: *a*, *b*, —, *m*, —, *z* —.

## 10. TRIDYMITE.

Small hexagonal plates of tridymite, 1 mm. in diameter and rarely larger, are rather plentiful in places. The forms present on two crystals which were partially measured are $c=\{0001\}$, $m=\{10\bar{1}0\}$, $i=\{32\bar{5}0\}$, $l=\{54\bar{9}0\}$, $r=\{30\bar{3}4\{$, $f=\}10\bar{1}2\}$. The first crystal measured was a twin.

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | ° | ′ | ° | ′ |
| $c \wedge \underline{c}$ | 35 | 06 | 35 | 18 |
| $c \wedge m$ | 90 | 02 | 90 | 00 |
| $m \wedge i$ | 23 | 20 | 23 | 25 |
| $m \wedge l$ | 26 | 39 | 26 | 20 |
| $c \wedge r$ | 54 | 48 | 55 | 03 |
| $c \wedge f$ | 43 | 54 | 43 | 39 |

## 11. ZINNWALDITE.

The occurrence of this lithia mica has already been mentioned by Collier,[a] who has very kindly furnished the writer the numerous samples collected. He gives as the exact locality: On Cassiterite Creek (extension or branch of Lost River), 5 miles from the coast and 8 miles east of Cape York, Alaska. It is stated to occur in a vein cutting through limestone. The bowlders showing the zinnwaldite are float, and in some of these there is a vein, about a centimeter thick, consisting almost entirely of zinnwaldite. Others show the zinnwaldite scattered through massive quartz. Cassiterite is associated with both modes of occurrence. As further associations Collier mentions fluorite, feldspar, calcite, tourmaline, topaz, pyrite, garnet, galena, wolframite, and malachite.

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[a] Collier, A. J., The tin deposits of the York region, Alaska: Bull. U. S. Geol. Survey No. 229, p. 20.

The zinnwaldite has a brown color, and occurs massive without any crystal boundaries. It fuses easily to a black mass, coloring the flame crimson. An incomplete quantitative analysis showed that the mineral is undoubtedly zinnwaldite. The small dikes before mentioned are bordered on one side by a rim of blackish green mica. This rim is about 1 mm. wide and is well developed only on one side of the dike, though it is present on the other side in smaller amount. Under the microscope the rim has a green and brown color and is uniaxial nearest the limestone, but the axial angle increases as the zinnwaldite is approached. In many places the two micas are intergrown. The difference in color from the green to the almost colorless zinnwaldite is very marked. The green mica is probably biotite, possibly somewhat altered to chlorite. Further description of the zinnwaldite is deferred for a later paper. Of the associated minerals, cassiterite and topaz occur in small crystals, a brief description of which follows. The association of cassiterite, topaz, and zinnwaldite is also noteworthy.

## 12. CASSITERITE.

The crystals of cassiterite vary in size, ranging from a thickness of less than 1 mm. to several cm. The large ones are opaque and black; the smaller ones are usually transparent and light brown in color, becoming nearly colorless at times. Twins are very common.

The large crystals usually show the simple combination of $a=\{100\}$, $m=\{110\}$, $e=\{011\}$, $s=\{111\}$, while the smaller ones usually are more complex. Two such small ones were measured, the first one a simple crystal, the second a twin. The forms noted are: $a=\{100\}$, $m=\{110\}$, $k=\{340\}$, $B=\{570\}$?, $r=\{230\}$, $h=\{120\}$, $e=\{011\}$, $s=\{111\}$, $f=\{231\}$, and the new prism $l=\{140\}$.

The following are the measured angles:

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | φ. | ρ. | φ. | ρ. |
| | ° ′ | ° ′ | ° ′ | ° ′ |
| *a*=100 | 0 00 | 90 00 | 0 00 | 90 00 |
| *m*=110 | 45 00 | 90 00 | 45 00 | 90 00 |
| *k*=340 | 36 35 | 90 00 | 36 52 | 90 00 |
| *B*=570 | 35 48 | 90 00 | 35 32 | 90 00 |
| *r*=230 | 33 37 | 90 00 | 33 41 | 90 00 |
| *h*=120 | 26 30 | 90 00 | 26 34 | 90 00 |
| *l*=140 | 13 56 | 90 00 | 14 02 | 90 00 |
| *e*=011 | 0 03 | 33 48 | 0 00 | 33 54 |
| *s*=111 | 45 01 | 43 35 | 45 00 | 43 33 |
| *f*=231 | 33 46 | 67 39 | 33 41 | 67 35 |

The prism $l=\{140\}$ was noted three times as a narrow face, measurements of which gave the following values for the $\phi$ angle:

| ° | ′ |
|---|---|
| 13 | 49 |
| 13 | 19 |
| 14 | 40 |

The dominant terminal form is {111}, though {011} becomes at times fairly large. {231} occurs on crystal No. 1 as minute faces and also as long, narrow line faces.

## 13. TOPAZ.

Two small crystals, about 2 mm. long, which were measured, showed the following forms:[a] $b=\{010\}$, $m=\{110\}$, $l=\{120\}$, $\pi=\{250\}$, $g=\{130\}$, $y=\{021\}$, $f=\{011\}$, $u=\{112\}$, $s=\{335\}$, $o=\{111\}$. The crystals were rounded and broken and gave, on the whole, rather poor signals. The measurements serve, however, to determine the forms.

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | φ. | ρ. | φ. | ρ. |
| | ° ′ | ° ′ | ° ′ | ° ′ |
| *b*=010 | 1 36 | 90 00 | 0 00 | 90 00 |
| *m*=110 | 61 59 | 90 00 | 62 08 | 90 00 |
| *l*=120 | 43 28 | 90 00 | 43 25 | 90 00 |
| *π*=250 | 37 00 | 90 00 | 37 07 | 90 00 |
| *g*=130 | 32 26 | 90 00 | 32 14 | 90 00 |
| *y*=021 | 0 38 | 61 44 | 0 00 | 62 20 |
| *f*=011 | 0 38 | 44 24 | 0 00 | 43 39 |
| *u*=112 | 61 56 | 45 39 | 62 08 | 45 35 |
| *s*=335 | 61 42 | 50 31 | 62 08 | 50 46 |
| *o*=111 | 63 10 | 63 00 | 62 08 | 63 54 |

The only terminal form on crystal No. 1 is $u=\{112\}$.

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[a] Goldschmidt's orientation, chosen in Winkeltabellen.

## 14. BOURNONITE.

An analysis of massive bournonite from Bogg's mine, Big Bug district, Yavapai County, Ariz., furnished for analysis by Prof. C. Palache, yielded the figures given below.

| | |
|---|---|
| S | 20.04 |
| As | 2.81 |
| Sb | 18.99 |
| Pb | 40.21 |
| Cu | 15.12 |
| Fe | .35 |
| Zn | .35 |
| Mn | trace |
| Insoluble | 1.67 |
| | 99.54 |

## 15. ANHYDRITE.

A deposit of crystallized anhydrite occurs in Texas, the exact locality of which is not known. The crystals are small, reaching a maximum of 1 mm. in length, and are tabular parallel to the macropinacoid. The base is present as a long, narrow face. The clinopinacoid is probably present, but the zone of brachydomes is very rounded and striated, no sharp faces occurring in this zone. Under the microscope the crystals show that the axial plane is parallel to $b=\{010\}$ and the acute bisectrix is normal to $a=\{100\}$. The interference figure is about as large as that of muscovite and is positive. An analysis was made of the mineral, to leave no doubt as to its identity.

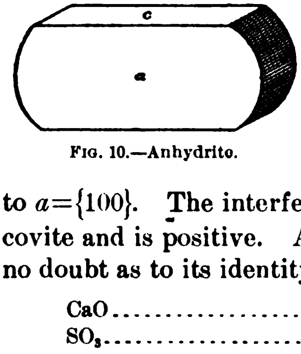

FIG. 10.—Anhydrite.

| | |
|---|---|
| CaO | 41.01 |
| $SO_3$ | 58.87 |
| Ign | .22 |
| Insoluble | .06 |
| | 100.16 |

## 16. GLAUCODOT.

A sample of cobalt ore from the Standard Consolidated gold mine, Sumpter, Oreg., consists of glaucodot, with pyrite and perhaps other sulphides, in a dark silicate rock. Imperfect crystals are abundant, though they usually show but one or two faces. Much of the ore is massive, and it is practically impossible to determine whether it is all glaucodot or in part cobaltite. One crystal was measured. The small crystal (about 1 mm. long) is orthorhombic, and shows the following forms: $m=\{110\}$, $b=\{010\}$, $e=\{101\}$, and a very much striated brachydome zone in which the following forms are probably present: *{014}, {013}, {012}.*

The following are the angles obtained:

| | Measured. | | Calculated. | |
|---|---|---|---|---|
| | φ. | ρ. | φ. | ρ. |
| | ° ′ | ° ′ | ° ′ | ° ′ |
| *b* =010 | 0 34 | 90 00 | 0 00 | 90 00 |
| *m*=110 | 56 10 | 90 00 | 55 34 | 90 00 |
| *e* =101 | 90 20 | 60 22 | 90 00 | 60 04 |
| *r* =014 | 0 34 | 15 33 | 0 00 | 16 34 |
| *q* =013 | 0 34 | 22 00 | 0 00 | 21 39 |
| *s* =012 | 0 34 | 29 00 | 0 00 | 30 46 |

Many other signals between these were seen in the striated brachydome zone, but they probably belong to no definite face.

The analysis gave the following results:

| | | Ratio. |
|---|---|---|
| As | 39.84 | 0.94 |
| S | 18.46 | 1.04 |
| Fe | 12.45 | 1.02 |
| Co | 20.23 | |
| Insol | 9.38 | .......... |
| | 100.36 | .......... |

Nickel and antimony were not determined, though they were shown not to be present in any quantity. The ratio of the cobalt to iron is 31:20 or 3:2, so that the formula may be written

$$Co_3\,Fe_2\,As_5\,S_5,$$

instead of

$$R\,As\,S,\ R=(Co,\,Fe).$$

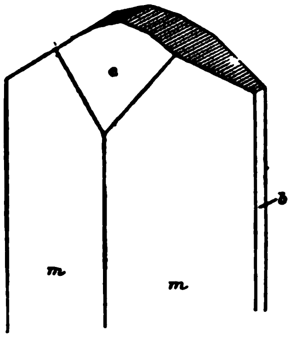

FIG. 11.—Glaucodot.

### 17. PYRITE.

A small group of pyrite crystals from Spanish Peaks, Colo., kindly presented to the writer by Mr. R. Sargent, showed, on examination, a rich combination of forms, especially in the pyritohedral zone. The faces are for the most part not striated and give excellent reflections. In size the crystals reach a maximum of about one-half centimeter in diameter. They were so closely crowded together that only a small portion of each crystal was developed; consequently, when the group was broken, a few fragments showed only one or two zones of faces.

Twenty-five forms are present on the twelve crystals measured. They are:

| | | | |
|---|---|---|---|
| *c* =100 | *α*=470 | *δ* =450 | *q* =112 |
| *A*=1.10.0 | *h* =350 | *ξ* =560 | *o* =111 |
| 𝔟 =180 | *C*=580 | *γ*=670 | *u* =221 |
| *f* =140 | *D*=570 | *d* =110 | *x* =123 |
| *a* =130 | *i* =340 | *K*=114 | *ψ*=124 |
| *e* =120 | *E*=790 | *m*=113 | *ω*=134 |
| *B*=590 | | | |

The average of the angles measured with the calculated angles are shown in the following table:

| No. | Letter. | Symbol. | Measured. φ. | Measured. ρ. | Calculated. φ. | Calculated. ρ. |
|---|---|---|---|---|---|---|
| | | | ° ′ | ° ′ | ° ′ | ° ′ |
| 1 | *c* | 100 | 90 00 | 90 00 | 90 00 | 90 00 |
| 2 | *A* | 1.10.0 | 5 49 | 90 00 | 5 43 | 90 00 |
| 3 | 𝔟 | 180 | 7 08 | 90 00 | 7 08 | 90 00 |
| 4 | *f* | 140 | 14 03 | 90 00 | 14 02 | 90 00 |
| 5 | *a* | 130 | 19 06 | 90 00 | 18 26 | 90 00 |
| 6 | *e* | 120 | 26 34 | 90 00 | 26 34 | 90 00 |
| 7 | *B* | 590 | 28 59 | 90 00 | 29 03 | 90 00 |
| 8 | *α* | 470 | 29 30 | 90 00 | 29 45 | 90 00 |
| 9 | *h* | 350 | 31 04 | 90 00 | 30 58 | 90 00 |
| 10 | *C* | 580 | 31 59 | 90 00 | 32 00 | 90 00 |
| 11 | *D* | 570 | 35 17 | 90 00 | 35 32 | 90 00 |
| 12 | *i* | 340 | 36 52 | 90 00 | 36 52 | 90 00 |
| 13 | *E* | 790 | 37 47 | 90 00 | 37 52 | 90 00 |
| 14 | *δ* | 450 | 38 39 | 90 00 | 38 39 | 90 00 |
| 15 | *ξ* | 560 | 39 48 | 90 00 | 39 48 | 90 00 |
| 16 | *γ* | 670 | 40 36 | 90 00 | 40 36 | 90 00 |
| 17 | *d* | 110 | 45 00 | 90 00 | 45 00 | 90 00 |
| 18 | *K* | 114 | 45 00 | 19 25 | 45 00 | 19 28 |
| 19 | *m* | 131 | 18 33 | 72 45 | 18 26 | 72 27 |
| 20 | *q* | 112 | 26 34 | 65 54 | 26 34 | 65 54 |
| | | 121 | 44 59 | 35 15 | 45 00 | 35 16 |
| 21 | *o* | 111 | 45 00 | 54 44 | 45 00 | 54 44 |
| 22 | *u* | 122 | 45 00 | 70 31 | 45 00 | 70 31 |
| | | 221 | 26 34 | 48 13 | 26 34 | 48 11 |
| 23 | *x* | 123 | 26 35 | 36 40 | 26 34 | 36 42 |
| | | 132 | 18 31 | 57 40 | 18 26 | 57 41 |
| | | 231 | 33 42 | 74 30 | 33 41 | 74 30 |
| 24 | *ψ* | 124 | 26 20 | 28 40 | 26 34 | 29 12 |
| 25 | *ω* | 134 | 18 18 | 39 37 | 18 26 | 38 19 |

Of the pyritohedrons *i* is the dominant form and is always well developed in large broad faces. *e*, *δ*, *ξ*, *γ* occur quite frequently as smaller faces, while **b**, *f*, *a*, *α*, *h*, are rarely present as very narrow faces. *c* is also narrow but never striated, always giving perfect reflections. The dodecahedron, *d*, is usually somewhat uneven. The octahedron, *o*, and *q*, *u*, and *x* occur commonly, while *k*, *m*, *ψ*, *ω* were observed but once each.

## 18. VANADINITE.

A specimen of vanadinite, doubtless from Yuma County, Ariz., shows a few crystals that possess a large number of forms. The crystals are 3 mm. long and 2 mm. thick, and present the usual appearance and habit of vanadinite. The zone *c m* is, however, very rich in forms. A number of vicinal forms were encountered that seem characteristic of the particular specimen. Sometimes the crystals are broken, or an apparently simple crystal may be composed of several ones not strictly in parallel position. In such cases, each zone *c m* was carefully adjusted by itself and the angles were measured. A continuous line of signals was practically never encountered, each face being distinct and giving a sharp signal.

The forms observed are quoted below.

Known forms:
$c=0001$
$a=10\bar{1}0$
$b=11\bar{2}0$
$h=21\bar{3}0$
$g=31\bar{4}0$
$\sigma=10\bar{1}3$
$r=10\bar{1}2$
$x=10\bar{1}1$
$\delta=50\bar{5}4$
$\gamma=40\bar{4}3$
$\alpha=30\bar{3}2$
$\beta=50\bar{5}3$
$y=20\bar{2}1$
$q=50\bar{5}2$

Known forms—Continued.
$z=30\bar{3}1$
$\pi=40\bar{4}1$
$s=11\bar{2}1$
$v=11\bar{2}2$
$i=21\bar{3}1$
$\xi=41\bar{5}4$
$l=32\bar{5}2$

New forms:
$d=53\bar{8}0$
$k=30\bar{3}4$
$o=70\bar{7}6$
$u=70\bar{7}5$
$A=70\bar{7}4$
$H=70\bar{7}2$

New forms—Continued.
$K=11.0.\overline{11}.3$
$L=10.0.\overline{10}.1$

Vicinal forms:
$26.0.\overline{26}.25$
$43.0.\overline{43}.40$
$19.0.\overline{19}.10$
$15.0.\overline{15}.7$

Doubtful forms:
$n=11.0.\overline{11}.10$
$B=90\bar{9}5$
$D=90\bar{9}4$
$\mathrm{E}=80\bar{8}3$
$\mathrm{G}=10.0.\overline{10}.3$
$\mathrm{F}=11.0.\overline{11}.4$

The measured and calculated angles for all these forms are:

| No. | Letter. | Symbol. | Measured. | | Calculated. | |
|---|---|---|---|---|---|---|
| | | Miller. | φ. | ρ. | φ. | ρ. |
| | | | ° ′ | ° ′ | ° ′ | ° ′ |
| 1 | *c* | 0001 | .......... | 0 00 | .......... | 0 00 |
| 2 | *a* | $10\bar{1}0$ | 0 00 | 90 00 | 0 00 | 90 00 |
| 3 | *b* | $11\bar{2}0$ | 30 00 | 90 00 | 30 00 | 90 00 |
| 4 | *h* | $21\bar{3}0$ | 19 09 | 90 00 | 19 06 | 90 00 |
| 5 | *g* | $31\bar{4}0$ | 13 54 | 90 00 | 13 54 | 90 00 |
| 6 | *d* | $53\bar{8}0$ | 21 34 | 90 00 | 21 47 | 90 00 |
| 7 | *σ* | $10\bar{1}3$ | 0 00 | 15 23 | 0 00 | 15 20 |
| 8 | *r* | $10\bar{1}2$ | 0 00 | 22 22 | 0 00 | 22 21 |
| 9 | *k* | $30\bar{3}4$ | 0 00 | 31 44 | 0 00 | 31 40 |
| 10 | *x* | $10\bar{1}1$ | 0 00 | 39 29 | 0 00 | 39 26 |
| 11 | — | $26.0.\overline{26}.25$ | 0 00 | 40 26 | 0 00 | 40 32 |
| 12 | — | $43.0.\overline{43}.40$ | 0 00 | 41 29 | 0 00 | 41 29 |
| 13 | *n* | $11.0.\overline{11}.10$ | 0 00 | 42 16 | 0 00 | 42 08 |
| 14 | *o* | $70\bar{7}6$ | 0 00 | 43 54 | 0 00 | 43 49 |
| 15 | *δ* | $50\bar{5}4$ | 0 00 | 45 47 | 0 00 | 45 47 |
| 16 | *γ* | $40\bar{4}3$ | 0 00 | 47 37 | 0 00 | 47 38 |
| 17 | *u* | $70\bar{7}5$ | 0 00 | 49 05 | 0 00 | 49 01 |
| 18 | *α* | $30\bar{3}2$ | 0 00 | 50 53 | 0 00 | 50 58 |
| 19 | *β* | $50\bar{5}3$ | 0 00 | 54 02 | 0 00 | 53 53 |
| 20 | *A* | $70\bar{7}4$ | 0 00 | 55 13 | 0 00 | 55 12 |
| 21 | *B* | $90\bar{9}5$ | 0 00 | 56 09 | 0 00 | 55 57 |
| 22 | — | $19.0.\overline{19}.10$ | 0 00 | 57 24 | 0 00 | 57 23 |
| 23 | *y* | $20\bar{2}1$ | 0 00 | 58 42 | 0 00 | 58 42 |
| 24 | — | $15.0.\overline{15}.7$ | 0 00 | 60 25 | 0 00 | 60 25 |
| 25 | *D* | $90\bar{9}4$ | 0 00 | 61 42 | 0 00 | 61 37 |
| 26 | *q* | $50\bar{5}2$ | 0 00 | 63 43 | 0 00 | 64 03 |
| 27 | *E* | $80\bar{8}3$ | 0 00 | 65 30 | 0 00 | 65 29 |
| 28 | *F* | $11.0.\overline{11}.4$ | 0 00 | 66 10 | 0 00 | 66 09 |
| 29 | *ζ* | $30\bar{3}1$ | 0 00 | 67 52 | 0 00 | 67 56 |
| 30 | *G* | $10.0.\overline{10}.3$ | 0 00 | 69 53 | 0 00 | 69 57 |
| 31 | *H* | $70\bar{7}2$ | 0 00 | 70 57 | 0 00 | 70 50 |
| 32 | *K* | $11.0.\overline{11}.3$ | 0 00 | 71 38 | 0 00 | 71 39 |
| 33 | *π* | $40\bar{4}1$ | 0 00 | 73 04 | 0 00 | 73 05 |
| 34 | *L* | $10.0.\overline{10}.1$ | 0 00 | 83 09 | 0 00 | 83 04 |
| 35 | *s* | $11\bar{2}1$ | 30 15 | 54 40 | 30 00 | 54 55 |
| 36 | *r* | $11\bar{2}2$ | 29 54 | 35 22 | 30 00 | 35 27 |
| 37 | *i* | $21\bar{3}1$ | 19 02 | 65 11 | 19 06 | 65 19 |
| 38 | *ξ* | $41\bar{5}4$ | 10 56 | 41 49 | 10 53 | 43 17 |
| 39 | *l* | $32\bar{5}2$ | 20 38 | 60 03 | 23 25 | 60 50 |

Following is a short description of the new forms, the $\rho$ angle only being given for the pyramids, $\phi$ being 0° 00′.

$d=\{53\bar{8}0\}$: Four faces of this new prism were noted, always as line faces.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas. $\phi$.... | 21 | 20 | Calc. $\phi$.... | 21 | 47 |
| | | 33 | | | |
| | | 37 | | | |
| | | 48 | | | |

$k=\{30\bar{3}4\}$: Only two faces of this form are present, both on the same crystal. The faces, though narrow, are broader than many of the others, and gave fairly good reflections.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 31 | 45 | Calc.... | 31 | 40 |
| | | 43 | | | |

$\{26.0.\overline{26}.25\}$: This and the following form are doubtless vicinal to $\{10\bar{1}1\}$, though they both occur a number of times. This one is present on four crystals, and the measurements show the following values:

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 40 | 24 | Calc.... | 40 | 32 |
| | | 26 | Calc. for $\{10\bar{1}1\}$ | 39° | 26′ |
| | | 36 | | | |
| | | 25 | | | |
| | | 19 | | | |
| | | 25 | | | |

$\{43.0.\overline{43}.40\}$: This vicinal form is present on three crystals and occurs nine times. The faces are small and the reflections fair.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 41 | 24 | Calc.... | 41 | 29 |
| | | 30 | Calc. for $\{10\bar{1}1\}$, | 39° | 26′. |
| | | 32 | | | |
| | | 16 | | | |
| | | 24 | | | |
| | | 35 | | | |
| | | 26 | | | |
| | | 35 | | | |
| | | 37 | | | |

$n=\{11.0.\overline{11}.10\}$: This form occurs on two crystals. The faces are narrow and the reflections but fair.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 42 | 24 | Calc.... | 42 | 08 |
| | | 20 | | | |
| | | 04 | | | |

The form can be only classed as doubtful.

$o=\{70\bar{7}6\}$: Occurring on three crystals, the following angles were measured:

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 43 | 24 | Calc.... | 43 | 49 |
| | | 53 | | | |
| | 44 | 06 | | | |
| | | 14 | | | |

$u=\{70\bar{7}5\}$: The reflections from the very narrow faces of this form were poor.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 49 | 17 | Calc.... | 49 | 01 |
| | | 16 | | | |
| | 48 | 42 | | | |

$A=\{70\bar{7}4\}$: This is the most prominent of the new forms, and occurs ten times as (relatively) broad faces on four crystals. The reflections were usually fairly good.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 54 | 52 | Calc.... | 55 | 12 |
| | | 49 | | | |
| | | 52 | | | |
| | | 48 | | | |
| | 55 | 35 | | | |
| | | 11 | | | |
| | | 38 | | | |
| | | 12 | | | |
| | | 24 | | | |
| | | 17 | | | |

$B=\{90\bar{9}5\}$: A very doubtful form, present as very narrow faces giving poor reflections.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 56 | 18 | Calc.... | 55 | 57 |
| | | 05 | | | |

$\{19.0.\bar{1}\bar{9}.10\}$: Another vicinal form, doubtless vicinal to $\{20\bar{2}1\}$.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 56 | 59 | Calc.... | 57 | 23 |
| | 57 | 18 | | | |
| | | 18 | Calc. for $\{20\bar{2}1\}$, | 58° | 42′ |
| | | 30 | | | |
| | | 19 | | | |

$C=\{15.0.\bar{1}\bar{5}.7\}$: Another vicinal form occurring on three crystals.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 60 | 13 | Calc.... | 60 | 25 |
| | | 38 | | | |
| | | 24 | | | |

$D=\{90\bar{9}4\}$: A doubtful form present on one crystal.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 61 | 53 | Calc.... | 61 | 37 |
| | | 30 | | | |

$E=\{80\bar{8}3\}$: A very doubtful form.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 65 | 30 | Calc.... | 65 | 29 |
| | 65 | | | | |

$F=\{11.0.\overline{11}.4\}$: A doubtful form.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 66 | 01 | Calc.... | 66 | 09 |
| | | 19 | | | |

$G=\{10.0.\overline{10}.3\}$: A doubtful form.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 69 | 54 | Calc.... | 69 | 57 |

$H=\{70\bar{7}2\}$: Though occurring but once as a small face, it gave a fairly good reflection.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 70 | 57 | Calc.... | 70 | 50 |

$K=\{11.0.\overline{11}.3\}$: Two faces of this form were noted.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 71 | 41 | Calc.... | 71 | 39 |
| | | 35 | | | |

$L=\{10.0.\overline{10}.1\}$: Two faces of this form are present on one crystal.

| | ° | ′ | | ° | ′ |
|---|---|---|---|---|---|
| Meas.... | 82 | 56 | Calc.... | 83 | 04 |
| | 83 | 21 | | | |

The average value obtained for $p_0$ is 0.8223.

## 19. GLOBULAR LEPIDOLITE.

Text-books on mineralogy do not mention the globular form for the lithia-mica lepidolite. Such globular forms have long been known for muscovite, and in fact are by no means uncommon for that species. While studying the literature of lepidolite in general two occasions were noted in which globular lepidolite was spoken of.

One was by vom Rath[a] in describing the lithia deposits of the island of Elba. On page 649 he says: "Zuweilen sind die (lepidolite) Blätter zu Kugeln gruppirt"—at times the (lepidolite) plates are grouped in spheres (=globular).

The second mention was found in Hamlin's[b] book on Mount Mica, Paris, Me., in which he says:

> On November 20, 1822, E. L. Hamlin sent to Professor Silliman, at Yale College, a box of minerals from the locality, with letters and catalogue as follows: * * *. In the catalogue Mr. Hamlin mentions lilac-colored mica as found in small globular concretions.

A large specimen of the pegmatite rock in front of Mr. Bowker's barn, at Mount Mica, shows a number of these spheres of lepidolite, several centimeters in diameter. At Auburn, Me., several specimens of globular lepidolite were found in the summer of 1904 by Mr. Wade, working on Pulcifer's ledge on Mount Apatite. Some of these were fairly large, one such specimen (by no means the largest) collected by the writer being 6 cm. in diameter and 2½ cm. thick. Here, at Mount Apatite, this globular form of lepidolite is abundant and forms one of its characteristic phases. It often has a whitish border (a few millimeters thick) of some undetermined mineral (probably a mica) surrounding it.

---

[a] Vom Rath, G., Die Insel Elba: Zeitschr. d. d. geol. Ges., vol. 22, 1870, p. 649.

[b] Hamlin, A. C., The History of Mount Mica, pp. 16-17.

## 20. LIBETHENITE.

Some minute crystals of libethenite, about a millimeter in diameter, from Yerington, Lyon County, Nev., proved to be well adapted for measurement. As the symmetry of the mineral has been regarded as orthorhombic by some writers and monoclinic by others, it was hoped that the measurement of these crystals might contribute something to the question. The results have been very satisfactory and show that the crystals measured are orthorhombic and not monoclinic.

The crystals were collected by Mr. D. Smith, to whom the writer is under obligations for the material investigated. Qualitative chemical tests showed the presence of copper, phosphoric acid, and water.

Some time after these measurements were made an article by Melczer[a] appeared on the symmetry and form of libethenite, in which he came to the same conclusion as the writer in regard to the symmetry of the mineral, his conclusion being based on a large number of measurements of Cornwall crystals. There can therefore hardly be any more question as to the orthorhombic character of libethenite.

The crystals measured are minute, less than a millimeter in diameter, but well formed, and the faces usually give very good and sharp signals. As the unit prism is present on all crystals, they were easily adjusted in true polar position and could be quickly and accurately measured with the two-circle goniometer.

Under the microscope the crystals always extinguish parallel to the crystallographic axes. No phenomena, such as twinning structure or twinning lamellæ, could be observed, either under the microscope or on the goniometer.

Seven of the most perfect crystals were removed from the matrix and measured. The forms observed are: $b=\{010\}$, $m=\{110\}$, $^*h=\{540\}$, $\delta=\{310\}$, $^*l=\{410\}$, $e=\{011\}$, $d=\{101\}$, $\{s=111\}$.

---

[a] Melczer, G., Ueber Libethenite: Zeit. f. Kryst., vol. 39, 1904, p. 288.

The angles measured and calculated[a] for the eight forms given are:

| No. | Letter. | Symbol. | Measured. | | | | Calculated. | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Miller. | $\phi$. | | $\rho$. | | $\phi$. | | $\rho$. | |
| | | | ° | ′ | ° | ′ | ° | ′ | ° | ′ |
| 1 | *b* | 010 | 0 | 01 | 90 | 00 | 0 | 00 | 90 | 00 |
| 2 | *m* | 110 | 46 | 11 | 90 | 00 | 46 | 10 | 90 | 00 |
| 3 | *h* | 540 | 52 | 26 | 90 | 00 | 52 | 28 | 90 | 00 |
| 4 | *δ* | 310 | 71 | 46 | 90 | 00 | 72 | 15 | 90 | 00 |
| 5 | *l* | 410 | 77 | 02 | 90 | 00 | 76 | 30 | 90 | 00 |
| 6 | *e* | 011 | 0 | 00 | 35 | 03 | 0 | 00 | 35 | 04 |
| 7 | *d* | 101 | 89 | 56 | 36 | 11 | 90 | 00 | 36 | 10 |
| 8 | *s* | 111 | 46 | 11 | 45 | 22 | 46 | 10 | 45 | 23 |

Besides these the prism $t = \{210\}$ seems to be present on one crystal, though the reflection was very poor and the measured angle varies several degrees from the calculated value.

Measured ($\phi$), 61° 55′, calculated ($\phi$), 64° 21′.

The new prism $h = \{540\}$ occurs but once, giving a fair reflection.

The new prism $l = \{410\}$ occurs five times, once as a broad face and the other times as a narrow face.

Goldschmidt does not give the form $d = \{101\}$ in his Winkeltabellen, but Melczer gives the form as present on his crystals, and it also occurs as long, narrow faces on the crystals measured by the writer.

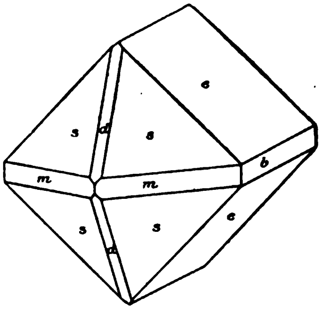

FIG. 12.—Libethenite.

In habit the crystals differ somewhat from that shown in Dana's system, inasmuch as on the Nevada crystals the pyramid *s* is large, equal in size to the brachydome, thus making the prism faces subordinate in size. The general appearance of these crystals is shown in fig. 12.

---

[a] Calculated angles taken from Winkeltabellen.

The combinations observed are shown in the following table:

| Crystal No. | *b* | *m* | *h* | *δ* | *l* | *e* | *d* | *s* |
|---|---|---|---|---|---|---|---|---|
| 1 | — | *m* | — | — | — | *e* | — | *s* |
| 2 | *b* | *m* | — | — | — | *e* | — | *s* |
| 3 | *b* | *m* | — | *δ* | *l* | *e* | *d* | *s* |
| 4 | *b* | *m* | — | — | — | *e* | *d* | *s* |
| 5 | *b* | *m* | — | — | *l* | *e* | *d* | *s* |
| 6 | *b* | *m* | — | — | — | *e* | — | *s* |
| 7 | *b* | *m* | *h* | — | *l* | *e* | — | *s* |

From all the good measurements, 27 values for $p_o$ and 24 values for $q_o$ were calculated, the average values being

$$p_o=.7308$$
$$q_o=.7014,$$

which agree very closely with those given by Goldschmidt, namely:

$$p_o=.7311$$
$$q_o=.7019.$$

The values obtained for $p_o$ and $q_o$ are as follows:

| $p_o$. | | $q_o$. | |
|---|---|---|---|
| Values. | Measurements. | Values. | Measurements. |
| .7295 | 2 | .6993 | 2 |
| .7297 | 2 | .7002 | 3 |
| .7301 | 1 | .7004 | 2 |
| .7302 | 1 | .7005 | 1 |
| .7303 | 3 | .7010 | 1 |
| .7304 | 3 | .7011 | 3 |
| .7305 | 1 | .7014 | 1 |
| .7308 | 1 | .7016 | 1 |
| .7309 | 1 | .7018 | 2 |
| .7312 | 3 | .7021 | 1 |
| .7314 | 2 | .7022 | 2 |
| .7316 | 3 | .7026 | 2 |
| .7319 | 2 | .7028 | 2 |
| .7321 | 2 | .7034 | 1 |
| Av., .7308 | 27 | Av., .7014 | 24 |

The difference in the $\phi$ and $\rho$ angles, for the forms, calculated from the two values, differ not more than a minute.

The average value obtained by Melczer is

$$p_0 = .7309$$
$$q_0 = .7027.$$

Taking the average of all three sets of values, we have

| $p_0$. | $q_0$. |
|---|---|
| .7308 | .7014 |
| .7311 | .7019 |
| .7309 | .7027 |
| Av., .7309 | Av., .7020 |

The axial ratio for libethenite then becomes

$$a : b : c = .9605 : 1 : .7020.$$

## 21. CRYSTALLOGRAPHY OF LEPIDOLITE.

A group of lepidolite crystals was collected by the writer in the summer of 1904 at a gem mine about 4 miles east of Ramona, on the stage road from Ramona to Julian, San Diego County, Cal. The mine was opened and worked for its gems, such as topaz, tourmaline, garnet, etc. With these are associated quartz, orthoclase in good crystals at times several feet long, muscovite, and, rarely, lepidolite. The minerals in question all occur in the loose dirt, filling "pockets" in a pegmatite dike.

The specimen of lepidolite under consideration consists of a group of intergrown crystals which often reach large size. The group itself is about 5 cm. thick and the individual crystals measure about 1 cm. across the base, though crystals with a diameter of 2 cm. are not rare. The crystals are also proportionally thick, the average being from 2 to 4 mm., though a few are somewhat thicker. They are transparent and of a very pale pink color, and, in the direction normal to the vertical axis and parallel to the base, of a slightly deeper tint. They fuse easily to a white enamel, coloring the flame an intense crimson. The side faces are usually plane, not rounded nor striated, and of a brilliant polish, giving excellent signals. It is, however, extremely difficult to secure a complete crystal from the group owing to the perfect basal cleavage, which will cause a crystal to split into a number of layers. This will also at times cause part of a crystal to become slightly displaced. In consequence of this difficulty most of the crystals measured are only parts of the original crystals, and two such pieces measured as separate crystals may, in reality, be parts of one and the same crystal.

As may be expected, the crystals of lepidolite are similar to those of muscovite. They also show, however, some marked differences. The interfacial angles are nearly the same and the crystals are naturally referred to the same axes as muscovite. There are three marked differences between these crystals and those of muscovite: (1) The rarity of twins, only one being found in twenty-one crystals; (2) the absence of the characteristic face of muscovite, $M=\{221\}$; and (3) the presence of the *a* face $\{100\}$, occurring on 10 out of 21 crystals. The crystals are not so striated as the green North Carolina muscovites, for instance, and in many cases the faces yielded perfect signals.

The common forms are $c=\{001\}$, $b=\{010\}$, $a=\{100\}$, $e=\{023\}$, $o=\{112\}$, $\mu=\{\bar{1}11\}$, and $x=\{\bar{1}31\}$. Besides these the following have been determined: $N=\{261\}$, $z=\{132\}$, $l=\{130\}$, and possibly several others, as $\{\bar{2}23\}$, $\{\bar{2}21\}$, $\{\bar{1}12\}$; also a number of forms which occur for the most part as broad faces, giving good reflections, but having anomalous indices, and which need further study.

The most frequent combination is *c b o u x*, after which comes *c b o u x e a*. Whenever the angle (001):(010) could be accurately measured it was found to be exactly 90° 00′ in every case, and the distribution of the faces also indicates monoclinic symmetry.

It was noticed that the angle between the same forms varied slightly on different crystals. The faces were smooth and highly polished, and the angles could easily be measured with an error not greater than 1′. It is suggested that this variation is real, and is due to the fact that lepidolite is an isomorphous mixture of two end products, and as the ratio of these two products varies, the crystallographic and physical properties of the mineral also vary. Further study on the possible correlation of the various properties of the mineral is under way.

This lepidolite belongs to the "second class" of Tschermak or the "brachydiagonal class" of Scharizer—i. e., the axial plane is parallel to the clinopinacoid and not normal to it. The trace of the axial plane was always parallel to one arm of the percussion figure.

The above is only a brief preliminary statement. The detailed crystallography of the mineral will soon be published in a paper on the mineralogy of this entire field of lithium minerals in southern California.

The writer's thanks are due Mr. Dan McIntosh, jr., of Ramona, the owner of the mine.

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DEPARTMENT OF THE INTERIOR
UNITED STATES GEOLOGICAL SURVEY
CHARLES D. WALCOTT, DIRECTOR

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# METHODS AND COSTS

OF

# GRAVEL AND PLACER MINING IN ALASKA

BY

CHESTER WELLS PURINGTON

WASHINGTON
GOVERNMENT PRINTING OFFICE
1905

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# LETTER OF TRANSMITTAL.

DEPARTMENT OF THE INTERIOR,
UNITED STATES GEOLOGICAL SURVEY,
*Washington, D. C., February 15, 1905.*

SIR: I transmit herewith a report by Mr. C. W. Purington on Methods and Costs of Gravel and Placer Mining in Alaska, and recommend its publication as a bulletin.

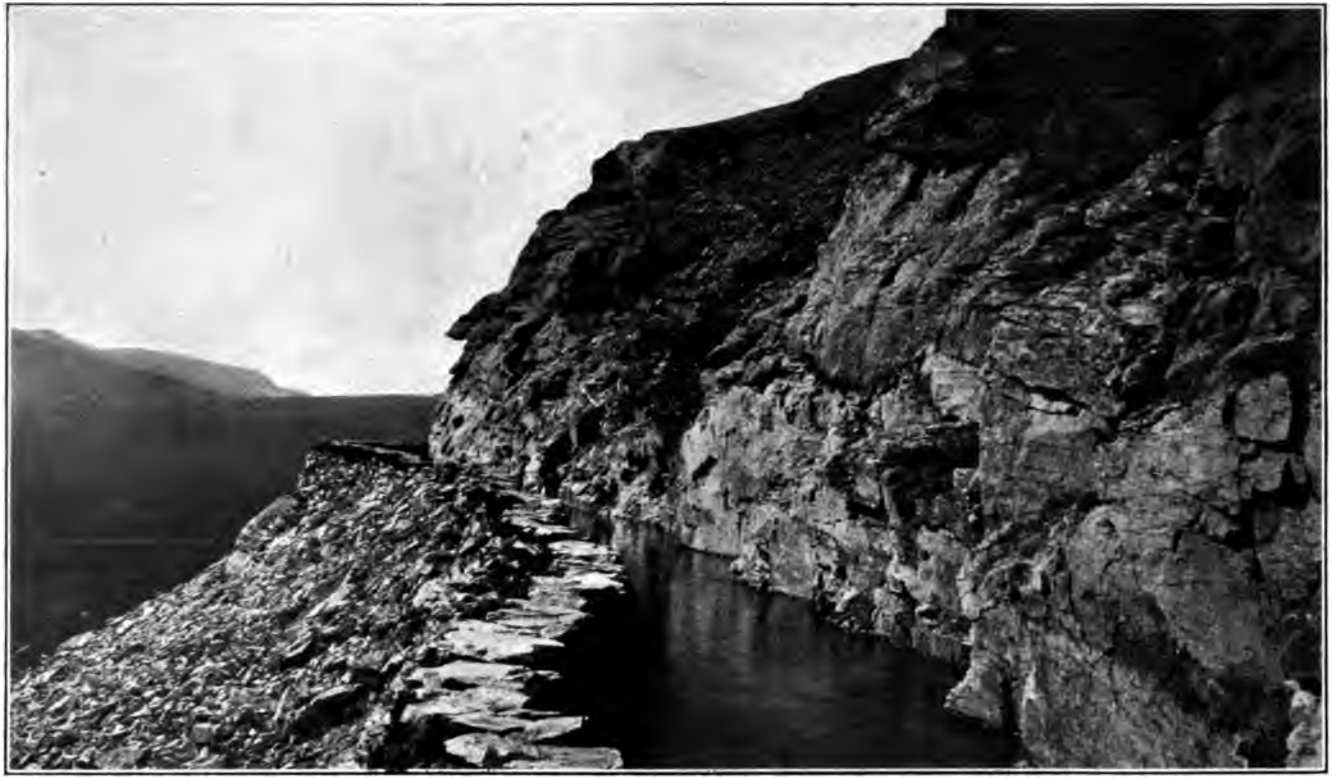
Placer mining in Alaska is affected by conditions which are not combined in any other field familiar to American miners. Hence new methods of operating must be substituted for those which are employed in other regions if the greatest success is to be obtained. In too many cases this fact has not been sufficiently recognized and many enterprises have therefore failed when they should have succeeded. These failures have tended to deter capital from entering a field which still presents many opportunities for profitable investment. Some of the factors which increase the cost of production of placer gold in Alaska, such as high rates for transportation, are changing for the better; others, such as the inadequate water supply and low-stream gradients which prevail in many localities, are permanent and must be met by properly applied methods of exploitation. It is these facts which led to the investigation so successfully carried out by Mr. Purington, the results of which are here presented.

It is believed that this report will stimulate the mining industry by making available, in compact form, the accumulated results of the experience of mining men in this field and by directing attention to those mining methods developed in older districts which appear to be adapted to the conditions prevailing in Alaska. While many of the figures presented, showing cost of labor, transportation, and fuel, are subject to fluctuation, with a general tendency to decrease, yet, as a comparative study of mining methods, this report will form a valuable permanent contribution to mining literature.

Very respectfully,

ALFRED H. BROOKS,
*Geologist in Charge Division of Alaskan Mineral Resources.*

Hon. CHARLES D. WALCOTT,
*Director United States Geological Survey.*



ROCK-CUT ON MIOCENE DITCH LINE, SEWARD PENINSULA.

# METHODS AND COSTS OF GRAVEL AND PLACER MINING IN ALASKA.

By C. W. PURINGTON.

## INTRODUCTION.

Owing to the fact that numerous requests have been received by the Geological Survey during the last three years for information regarding the cost of operating gold-bearing alluvial deposits in Alaska and the best means of working the claims in the various districts, the preparation of a report embodying such information, in so far as is possible from data at present obtainable, was decided on.

I was requested, in the spring of 1904, by Mr. Alfred H. Brooks, geologist in charge of the division of Alaskan mineral recources, to proceed to some of the most important placer fields of Alaska during the open season of 1904 and to collect data for an economic report. I was asked also to visit the neighboring British fields of Atlin and Klondike for the purpose of comparison. Mr. Sidney Paige was assigned as my assistant for the field and office duties, and has rendered most efficient aid during the entire progress of the work.

In the north the Juneau, Atlin, Klondike, Birch Creek, Fairbanks, and Seward Peninsula districts were successively visited, and on the return a short stay was made at the gold-dredging field of Oroville, Cal. (See Pl. II, p. 14, general route map.) Data concerning the districts not visited—the Porcupine, Chistochina, Cook Inlet, Fortymile, and Rampart, and remote parts of Seward Peninsula—have been collected from reliable sources, especially from members of the Geological Survey who have made investigations in those portions of Alaska.

## ITINERARY.

A start was made from Seattle May 26. Five days, from May 30 to June 3, were spent in the vicinity of Juneau, Alaska, where the placers of Silver Bow basin and the power plant of the Alaska-Treadwell mines were inspected. Six days, June 11 to 16, inclusive, were spent in the Atlin district of British Columbia, where ten plants were visited on Pine, Willow, Boulder, Spruce, and McKee creeks and Gold Run. Twenty-two days, June 25 to July 16, inclusive, were spent in the Klondike district of the Yukon Territory, where 32 operations were visited on Gold Run, Sulphur, Upper Dominion, Bonanza, Eldorado, Hunker, Last Chance, Bear, and Gold Bottom creeks. Three days, July 17 to 19, inclusive, were spent in the vicinity of Eagle, Alaska, two operations on American Creek being visited. At Circle the regular route of transportation via Yukon River was departed from. On July 22 the party started west with a pack train and reached Deadwood Creek, in the Birch Creek district, on July 24. This day and the next were spent on Deadwood Creek, where 6 operations were inspected. On July 26, 5 operations on Mammoth and Mastodon creeks were visited. On July 27, 4 operations on Eagle Creek were visited.

The route was then pursued westward over the ridge trail to Fairbanks Creek. From August 1 to 5, inclusive, nearly all the operations in the recently opened Fairbanks district were visited, including 35 operations on Fairbanks, Chatham, Cleary, Twin, and Pedro creeks. On Fairbanks Creek a junction was made with Mr. L. M. Prindle's party, and it is due to the fact that the two parties worked together in the Fairbanks district that all the operations were visited in the few days available. During the few succeeding days as much information as possible was collected in the town of Fairbanks concerning the new gold finds of the Tanana, and my party then proceeded via the Tanana, Yukon, and St. Michael to Nome, arriving there August 20. From August 21 to September 4, inclusive, 21 operations were inspected on Anvil, Glacier, and Dexter creeks and Grass Gulch, and on Newton, Peluk, and Dry creeks.

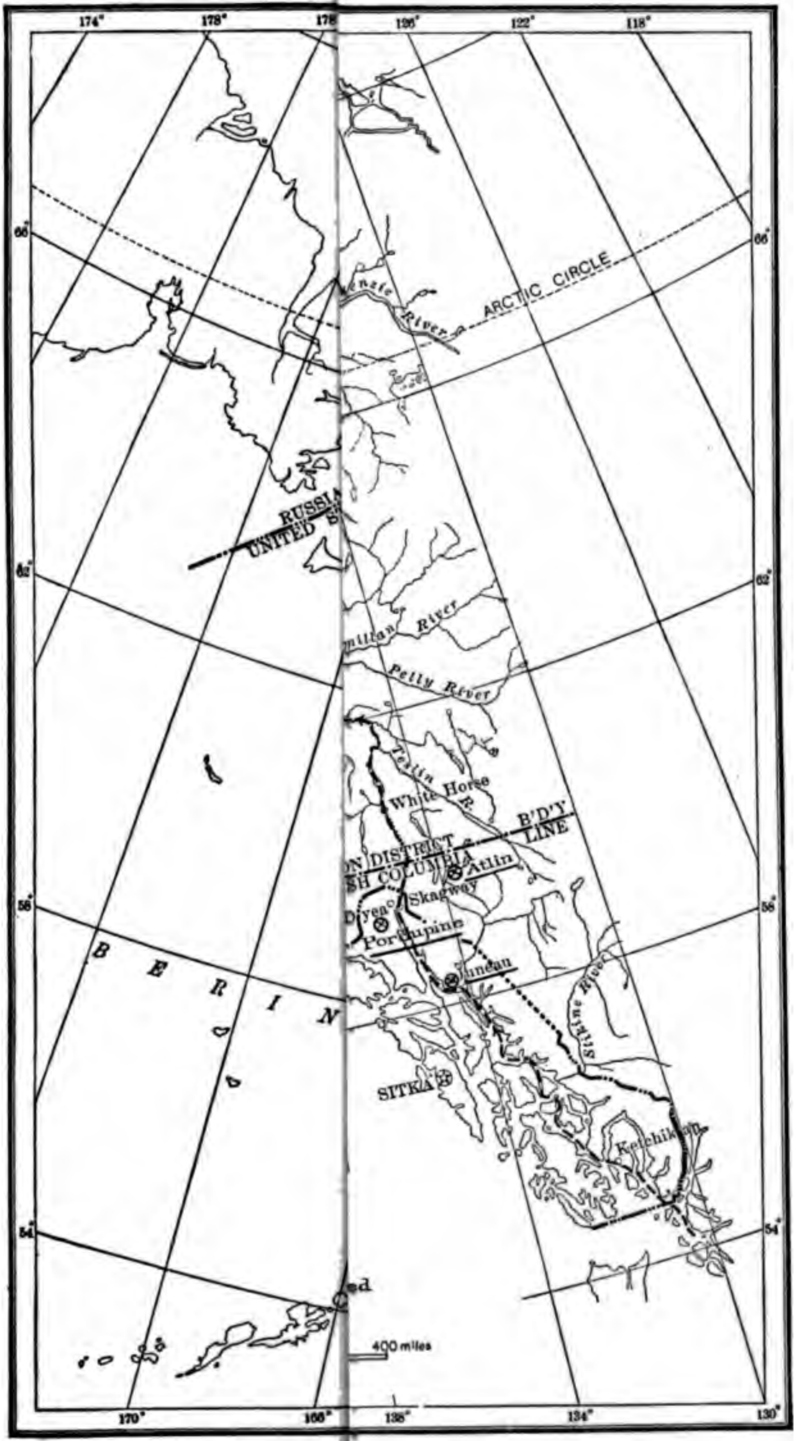
On September 5 the party proceeded to Ophir Creek via Cheenik and Council, and from September 5 to 17, inclusive, inspected 29 operations on Ophir, Crooked, Warm, Gold Bottom, Penelope, and Big Hurrah creeks and Solomon River.

After returning to Nome a week was consumed in completing at the offices of the mining companies the data already partially obtained on the creeks and in gathering all possible information concerning operations in portions of Seward Peninsula not visited. The party arrived in Seattle October 2, and after a few days spent at Oroville, Cal., returned to Washington.

## FIELD METHODS.

Every operation in a district was not visited, but enough representative claims were examined to give a fair estimate of the cost, utility, and adaptability of each method of mining.

For assistance in obtaining the data necessary a blank, a copy of which is given on pages 17–20, was sent to all the placer miners of Alaska whose addresses were known. This blank was accompanied by a circular letter, and in many cases by a personal letter, explaining the purpose of the investigation. The fact that replies have been received from all parts of the Territory, in many cases from the most remote


174°
178°
178
126°
122°
118°
66°
62°
58°
54°
ARCTIC CIRCLE
River
RUSSIA
UNITED ST
River
Pelly River
Teslin R.
White Horse
B'D'Y LINE
ON DISTRICT
SH COLUMBIA
Atlin
Skagway
Dyea
Porcupine
Juneau
Stikine River
B E R I N
SITKA
Ketchikan
ed
400 miles
170°
166°
138°
134°
130°

camps, shows that the mine operators take an interest in contributing to the preparation of the work in hand.

At the operations visited by myself or by Mr. Paige more detailed information was collected regarding costs, the handling of water, the setting up of the plant, and the mechanical appliances. Forms 1, 2, 3, 4, or 5 (pp. 17–25) were used, according to the character of the operations. Photographs were taken of all the operations visited. Over 400 excellent photographs have been obtained by Mr. Paige, some of which are reproduced in this report.

In each district a study has been made of the water resources, the most important factor in placer and hydraulic mining, and of the cost of building and maintaining ditches, flumes, and storage reservoirs for the purpose of using water under pressure. The important development of ditch building in Seward Peninsula has been especially considered.

The quantity and quality of timber in the mining districts has been considered with reference to its use for fuel, for building flumes and sluice boxes, and for timbering drift mines. It is an unwelcome fact that the important placer districts of the Northwest do not contain a great amount of timber, and the mining communities of the interior must use with discretion the scant supply of native timber at their command.

The questions of wages and cost of living have been considered from the standpoint of both the laborer and his employer. Temporary conditions govern wages in Alaska to a notable degree, and the table of wages applying in 1904 must be understood to have only present value. While the rate of pay for miners appears attractively high, the cost of living and of transportation to and from the country must be considered, as well as the shortness of the active season and the low rate of winter wages. Alaska presents a small field for those whose aim is limited to the earning of a day wage. On the other hand, men who are willing to work for wages during the summer, and actively and intelligently prospect during the winter with the money so earned, have always the chance of finding and locating a pay streak for themselves.

The main object has been the determination of the cost and most expeditious means of getting out the auriferous material—gravel, sand, or bed rock; of the cost of obtaining its valuable contents; of the best methods of hydraulic mining; of the cost of removing overburden under each set of conditions, and of handling the tailings; of the capacity and cost of installing mechanical methods; of the cost, capacity, and adaptability of the methods employed to thaw frozen gravel; and of the most feasible method of mining in little-developed districts.

Care has been taken to avoid the publication of figures which were accompanied by the request that they remain confidential. It may, in general, be said that the detailed data furnished have been used mainly for the purpose of compilation, and have not been herein specifically transcribed.

In addition to matters relating directly to the operation of placer mines, attention has been given to means and cost of freight and passenger transportation. Whoever contemplates engaging in mining in the northwest portion of America, especially in the interior Alaska country, can not devote too much attention to the study of the cost of transportation of supplies. For the benefit of those who must depend on local supply points in the north for their needs, a table is given showing present costs, at each principal point, of the articles most necessary to the miner.

As regards transportation facilities, care has been taken to collect and embody in this report all available data and statistics relating to the construction of wagon roads in the Northwest. The neighboring fields of British Columbia and the Yukon Territory, in Canada, have afforded excellent opportunity to obtain costs and methods of road construction applicable to the Alaska interior. Recommendations regarding the most practical and serviceable routes for road construction in Alaska have been made in a previous report by Mr. Alfred H. Brooks.[a] The need of these roads can not be urged too strongly, and the expenditure of $1,000,000, as suggested by Mr. Brooks, would be many times repaid by the resulting development and increased gold production. Already over 300 miles of wagon road have been built by the Canadian government in the Yukon Territory and the Atlin district of British Columbia, while over 600 miles of sled roads have been made in the Yukon Territory. The fact that in the summer wagons and vehicles of all descriptions, and even bicycles, may be seen daily about Dawson, the Klondike creeks, and Atlin, in British Columbia, while the winter roads in Canadian territory afford continuous easy routes for horse sleds down the Yukon to Dawson, is evidence of the success of the Canadian road-building enterprise. On the other hand, there are in Alaska less than 50 miles of well-built wagon roads, and these have been constructed by private enterprise.

No attempt has been made to give a detailed account of the topographic and geologic conditions in the districts visited, except where such conditions have a direct bearing on the method to be employed in working the ground. The Geological Survey has already published many exceedingly valuable reports and maps, most of which are distributed free of charge. A list of the Survey publications on Alaska is given at the end of this bulletin. Any prospector who is unprovided with the reports concerning the region in which he is *working* lacks an important part of his equipment.

---

[a] *Placer mining in Alaska*: Bull. U. S. Geol. Survey No. 225, 1903, p. 57.

It should be distinctly understood that the purpose of the expedition on which this report is based was not the sampling of gold-gravel deposits. Sampling is the business of the mining engineer acting in a private capacity, and is neither practicable nor permissible for officers of the Federal Government.

Information concerning the tenor of the gravels worked was frequently given by operators whose claims were visited, and such statements have been considered in the estimates of the average tenor in gold of the gravels.

On the following pages are reprinted the circular letter and the forms which were sent to miners in Alaska:

### *Circular letter sent to placer miners.*

Your kind cooperation is earnestly requested in the preparation of the report on the placer mining methods of Alaska, which has been undertaken by this Office under the supervision of Mr. Alfred H. Brooks, geologist in charge of division of Alaskan mineral resources.

Mr. Chester W. Purington has been detailed to investigate some of the principal placer districts of the Northwest during the coming summer, and will personally obtain as much information as possible. It being impracticable, however, to visit each operator, you are requested to assist in the preparation of this report by furnishing such data concerning your placer and the methods employed in operating it as are asked for on the accompanying blank form. The information furnished will materially aid the investigation and will insure the inclusion of all the districts in the final report, which it is expected will be distributed in 1905.

Methods of operating are improved and rendered more effective by a community of interest, and while data furnished by you and made use of in the report may assist some other operator, you also may derive benefit from it through the suggestions of others.

The results already attained in placer mining in Alaska and neighboring territory will be made use of for the purpose of outlining the most advisable methods to be followed in each of the principal placer districts. Especial attention will be given to gold dredging, hydraulic placer mining, the employment of elevators for obtaining artificial grade, and the determination of the best methods for the thawing of frozen gravel. A special feature of the report will be a comparison of costs of operation by various methods in all parts of the placer fields. It is evident that for carrying out the plan full data are necessary concerning the character of the ground, the local conditions, and the methods at present in use.

As the value of these reports depends largely upon the promptness of their publication, an early reply is earnestly requested. Please answer the questions on the schedule as far as possible. If you can not give the exact figures in all cases, make the replies as approximately correct as possible.

A copy of the report will be sent to each operator furnishing a statement. An addressed envelope is inclosed, which requires no stamp.

### *Form No. 1, sent to gold-placer miners of Alaska.*

It is hoped that even in cases where the operator does not care to give full data concerning his property and operations he will furnish at least a part of the information requested. Very few of the facts obtained will be embodied directly in the report, but from the sum of them will be drawn conclusions and suggestions which it is hoped will be of value to the gold-mining community as a whole.

1. What is the name and post-office address of firm, corporation, or individual making this report?
2. What is the name of mine?
3. What is the location of mine, and in what district? (NOTE.—Give name by which district is commonly known, such as "Birch Creek," "Ophir Creek," etc.)
4. By whom is the property owned?
   Address:
5. By whom is the mine managed?
   Address:
6. (*a*) How long has the mine been operated, and what was date of opening?
   (*b*) How many ounces of gold has it produced?
   (*c*) What was the gold worth per ounce?
7. What is the character of the deposit? (Creek, bench, river bar, seabeach, or gravel plain.)
8. (*a*) How many acres does it contain?
   (*b*) What are its approximate dimensions?
9. How many men are employed, and how much are they paid?
10. How many shifts are worked, and how many men on each shift?
11. Between what dates is gravel extracted?
12. Between what dates is it washed?
13. (*a*) Is there timber on the property?
    (*b*) If so, how much, and of what kinds and average size?
    (*c*) What does sawed lumber cost per thousand?
    (*d*) Where is sawed lumber obtained?
14. (*a*) If gravel is in a stream bed, what is the grade of the stream per mile, in feet?
    (*b*) How many miner's inches or cubic feet of water are used in mining operations, and for how many months in the year?
15. How much, in dollars or cents, must the gravel contain per cubic yard to pay?
16. How far below the surface does the pay gravel lie?
17. Will you please describe a section of the overburden, giving the succession and thickness of turf, soil, sand, round or angular gravel, clay, etc.?
18. (*a*) Does pay lie directly upon bed rock? (*b*) If not, upon what does it rest, and what is beneath?
19. How wide and thick is the pay?
20. Does it follow any definite channel or channels?
21. Is there more than one pay streak?
22. Is the pay gravel sandy or clayey?
23. (*a*) Is the gravel rounded or angular?
    (*b*) What is the usual diameter of the stones?
    (*c*) What per cent are over 6 inches in diameter?
    (*d*) What per cent are over 18 inches in diameter?
24. (*a*) Is ground frozen?
    (*b*) If so, to what depth?
25. (*a*) Is bed rock hard or soft?
    (*b*) What kind of rock?
    (*c*) If slate, or thinly bedded rock, what is its angle or dip?
26. (*a*) Is bed rock excavated in mining?
    (*b*) To what depth?
27. (*a*) Do you get coarse or fine, flaky or shot gold?
    (*b*) Are nuggets found?
    (*c*) Of what size?

28. (*a*) Is gold smooth or rough?
    (*b*) Rusty or bright?
    (*c*) Flat or chunky?
29. (*a*) Does nature of gold vary from grass roots to pay or in different parts of the pay streak?
    (*b*) If so, in what way? Please describe.
30. (*a*) What amount of black sand is caught per cubic yard of gravel washed?
    (*b*) What amount of other materials?
31. (*a*) Are other valuable minerals caught besides gold?
    (*b*) If so, what are they and in what amount?
32. Under which of the following heads do your operations come, and will you briefly describe the important features? (Operations: tundra, beach, creek, or bench mining.)
33. How many cubic yards of gravel are washed per day during the season?
34. How many cubic yards of overburden are stripped during the season?

### PROSPECTING GROUND.

1. (*a*) Has ground been prospected by pits or by drill holes?
   (*b*) If by pits, what was the size and depth?
2. (*a*) How many were dug?
   (*b*) What did they cost?
3. (*a*) Did they need timbering?
   (*b*) If so, how closely and how heavily?
4. If ground was prospected by drilling, what size, depth, and number of holes were used?
5. (*a*) What was the cost per foot?
   (*b*) What was the cost of equipment?
6. What difficulties were encountered?
7. How did the results from mining compare with the probable results as indicated by prospecting?

### HYDRAULIC OPERATIONS AND EQUIPMENT.

1. How much water is used, and how is it distributed in the operations?
2. What is the length, capacity, and grade of ditches, and head and tail flumes?
3. What is the length and size of pipe lines, and of what is pipe made?
4. Will you kindly describe giants, as to size, make, capacity, effectiveness, etc.?
5. How are bowlders handled?
6. What and how much material is consumed, such as lumber, quicksilver, powder, etc.?
7. What is the effectiveness of a giant upon frozen gravel?
8. How much gravel is moved in 24 hours, and what does it cost?
9. Are elevators used, and if so, what has been your experience with them?
10. How much water do they use?
11. Of whose make are they, and will you kindly describe them?
12. How large stones will they handle?

### DREDGING.

[NOTE.—If blueprints or photographs are furnished, they will be returned if desired.]

1. Will you kindly give a general description of dredge: Maker, type, size, estimated capacity, cost, and some details of construction; especially of new features that have proved advantageous.
2. How are bowlders handled?

3. How deep will buckets or dipper dig?
4. What is capacity of buckets?
5. What are arrangements for moving dredge and holding it in place?
6. How is gravel sized?
7. Are plates or quicksilver used?
8. What is the life of wearing parts?
9. What kind of power and engines are used?
10. What fuel is used, and what is its cost?
11. What type of pumps are used, and what is their capacity?
12. How many cubic yards of gravel have been handled in a given time from past record?
13. What special difficulties are encountered?
14. What is cost of dredging under local conditions?
15. Is frozen ground encountered, and if so, how does dredger work in it?

In case of special operations not covered by the above summary, will you kindly give brief descriptions, especially of apparatus which has proved successful in handling placer gravel under northern conditions?

The following space is left blank in order that any special description of the property may be given.

You are requested to fill out this schedule as soon as possible and return in the inclosed envelope which requires no stamps.

Forms 2, 3, 4, 5, 6, contained the first thirty-four questions in form 1, and the questions in form 1 regarding prospecting ground. In addition the following data were asked for, a separate form being used according as the operations came under the heads of hydraulic mining, dredging, open cutting, or drift mining.

### *Form 2.—Hydraulic mining.*

Size of head-gate.

Dimensions, grade, length, capacity, and cost of head ditch.

Cost of maintaining.

Length of flume, if any.

Dimensions, grade, and size of lumber in bottom, sides, and battening; size and length of posts and sills in head flume; number of collars to box.

Amount of trestling and cost of same. Size of sills, posts, and stringers.

Is flume or ditch covered to prevent freezing? Describe method.

Number of relief gates in ditch line.

Are inverted siphons used? If so describe.

Amount of water available.

Size pressure box and size lumber used.

Head of water in feet.

Length of pipe line from pressure box.

If more than one pressure box or pipe line, describe.

Size, gage, cost, and nature of pipe employed.

Where obtained.

Kind of joints.

Kind and number of air valves.

Method of setting and unsetting pipe.

Size of pipe at giant or giants.

Is water diverted for other purposes beside giant?

If so, how much, and for what purposes?

Number and angles of elbows in main pipe line.

Size, make, and cost of giant or giants used.
Size nozzle used, with or without deflector.
How often is giant moved?
What height of bank is operated on?
What modifications adopted in system of piping owing to frozen gravel?
Does giant clean bed rock?
If not, how is bed rock cleaned?
How much gravel is moved per shift?
If bankhead or by wash water used, how much?
How are bowlders handled, derricking or shooting?
Cost of handling bowlders.
Size, capacity, and cost of derrick and plant; cost to operate.
At what distance is head box of tail sluice from giant?
Dimensions, grade, and length of boxes in tail sluice.
Number of boxes.
Size of lumber in bottom, side, and lining boards.
Method of making sluice box, size of posts and sills.
Cost and time to make and put in place one box.
Life of lining boards.
Largest stones put through sluice.
Average cost of shooting bed rock to keep sluice on grade.
Kind of riffles employed in the sluice—distribution.
Cost and life of same—preference of riffles.
Amount of quicksilver used.
Amount of quicksilver consumed.
Are undercurrents used? Description, number, distribution, area of each, grade, results.
Per cent of gold saved in different parts of sluice.
Per cent of gold saved in undercurrents.
Per cent of gold lost.
Is gravel forked in the sluice?
Is dump moved after leaving end sluice box?
If so, by whāt power and to what distance?
Are tailings impounded? If so, for what reason?
Method and cost of impounding.
Cost of breaking down, moving, and washing gravel per cubic yard.
Improvements contemplated or suggested.
If hydraulic elevator is used—make, size, weight, and cost of elevator.
Depth of sump.
Effective lift of elevator.
Pressure necessary to obtain lift.
Size of throat, upcast pipe, and nozzle.
Weight, thickness, material, life, and cost of throat.
Method of joining upcast pipe.
Shape, material, and life of hood.
Largest stones handled.
Amount of water handled by elevator.
Amount of gravel handled by elevator.
Amount of water fed to nozzle.
Are auxiliary water lifts used?
Is any method beside piping used for feeding gravel to elevator? What method.
Describe sluice boxes before elevator, if any—number of boxes, grade, dimensions, riffles.
Describe tail sluice—length, grade, dimensions, riffles.

Per cent of gold caught before and after elevator.
Frequency, time, and cost of moving elevator.
Difficulties encountered in its use.

### *Form 3.—Dredging.*

Make and cost of dredge.
Estimated and actual capacity of dredge in cubic yards per twenty-four hours.
Is dredge in a river or in self-made pond?
Depth to which buckets dig, and height of bank above water level.
Where was hull constructed and of what material?
Feet of lumber in hull and cost of hull.
Dimensions and draft of hull, loaded.
Weight of machinery.
Power used, nature, amount, cost per day.
Height, size, method of bracing, and inclination of gauntrees.
Size, weight, position, and description of spuds, if used.
Size, number, position, and use of winch drums.
Position of lever house.
Size, number, and length of cables.
Does dredge dig on spud or head line?
Length, width, weight, material, of digging ladder.
Material and weight of tumblers.
Number, capacity, arrangement, weight, shape, and cost of buckets.
Material, life, and cost of bucket lips, pins, and bushings.
Are teeth used on any buckets?
How far in front of bow will buckets dig?
What size of stones are handled?
Do buckets clean bed rock?
Describe save-all, if used.
Is nozzle used for cleaning buckets?
Is trommel or shaking screen used? Preference.
Dimensions and life of trommel; size holes.
Is side chute used for large stones? Describe grizzly.
Is sluice box with auxiliary float used? If so, describe.
Is stacking ladder used? Length, how driven, steel-link belt, or rubber belt, width.
Height and distance from stern stacked.
Life, efficiency, and cost of stacker belt.
Treatment of fines, area, grade, and description of tables; kind of riffles most successful, amount of quicksilver used.
Is sand pump used? How often?
Clean up, how often? State method used and time consumed.
Size, make, and cost of pump used for supplying sluice water; height of lift.
Distribution of power; how much to each operation.
Size and capacity of boilers and engines or dynamos used; describe connections pulleys, and clutches.
Special difficulties owing to northern conditions.
Most frequent breakages.
What kind of ground does dredge work best in?
What kind of ground does dredge have most difficulty in?
If in pond, is water pumped in? How much? From what source?
Cost per cubic yard of digging and washing gravel.

### *Form 4.—Open cut (not worked by hydraulicking).*

Depth of overburden stripped.

Method of stripping; by ground sluicing, hydraulicking, scraping, or booming.

Cost per cubic yard of stripping; amount stripped in a given time.

Source of water.

Amount of water in miner's inches.

Length of ditch or flume and dimensions.

If water is pumped for sluicing, kind of pump used and cost of pumping per cubic yard of gravel handled.

Dimensions and grade of sluice boxes and number used.

Is mud box used? Dimensions.

Disposal of large stones.

Size of lumber used; one or more strings of boxes.

System of joining and setting up boxes.

Kind of riffles, cost and life.

Dimensions of pit worked with one set-up, frequency of new set-ups, and time consumed in changing.

How is seepage water handled, by pump or drain?

Dimensions, length, grade, capacity, cost, and method of constructing drain; difficulties with drain.

Is China pump used? Capacity; cost of pumping.

Is power pump used? Capacity; cost of pumping.

Describe system of dams, if used, method of construction and cost.

How are tailings handled? If scraped by horses, cost of scraping.

Per cent of gold saved in boxes, if known.

The following special queries apply to open-cut work:

If men shovel into boxes—
- To what height do men throw?
- Are platforms used?
- Depth of ground shoveled in.
- Amount of ground handled per shift per man in cubic yards.

If horse scrapers scrape into boxes—
- How many cubic yards per day will one team scrape into boxes?
- Cost per day of team and driver.
- Per cent of gold saved.

If steam scraping is employed, in handling pay or tailings—
- Describe plant and cost, cost of operation, capacity, and difficulties encountered.
- Amount and kind of fuel used.
- Number of men required to scraper.
- Distance to which material is conveyed, and height elevated.

If derricking is employed—
- Describe capacity of plant, give cost, system of operating, amount and distribution of power and labor.

If inclined track and car system is used—
- Number and capacity of cars handled per shift; number and length of tracks in use.
- Length of tram to bottom of incline.
- Number of men to a car.
- Length of incline and vertical height of dumping platform above bed rock.
- Amount of power to elevate; system of elevating, automatic devices used.

If steam shovel is used—
- Size, make, capacity, and cost of shovel.
- Power used, amount, and how generated.
- *Capacity of dipper.*

If steam shovel is used—
  Length of boom.
  Weight of machine.
  On track or skids.
  Height of bank on which it will operate.
  Does it clean bed rock, or do men clean bed rock after it?
  Efficiency in digging the material at hand.
  Does it dump into cars or sluice boxes?
  Is conveying system equal in capacity to shovel?
  Difficulties encountered.
  Cost per cubic yard of digging.
  Elevation to which gravel is hoisted in cars, power necessary for hoisting.
  Describe track system for returning empty cars.

If more than one of the above systems is in use for working open cuts on the same claim, give comparative efficiency and cost.

*Form 5—Drift mining.*

Dimensions of shaft.
Cost and rate of sinking per foot.
Is shaft timbered?
Method of timbering shaft.
Depth of shaft to bed rock.
Unfrozen gravel—
  Method of drifting.
  What width of face is carried?
  Are pillars left, what dimensions?
  Height of posts, length of caps.
  Are false sets used?
  Method of filling or caving worked-out ground.
  Are timbers recovered?
Frozen gravel—
  Rate and cost of sinking shaft by steam point.
  Diameter of steam pipe entering shaft.
  Length, diameter, and cost of points.
  Pressure of steam, size of cross-head.
  How far apart are points placed in face?
  Is hot water used in starting points? Preference.
  Horsepower per point.
  How long are points left in face?
  Cubic yards of gravel thawed by each point in given time.
  System of mining and hoisting.
  If gravel thawed by hot-water hydraulicking, describe method.
  Size, capacity, and cost of pump.
  Position of pump.
  Temperature of and method of heating water.
  Amount of water used.
  Size nozzle used.
  Cubic yards of gravel thawed per shift.
Boiler—kind, make, cost.
Kind, amount, cost of fuel burned per day.
Distribution of power, to thawing, to hoisting and conveying.
If gravel thawed by wood fires, describe method; give cost and capacity.
Describe system of tramming, wheeling, or skidding to shaft.
Describe system of hoisting.
Is self-dumping carrier used? Kind, efficiency.

**To what distance and height above the collar of shaft does it transport the gravel?**
**Capacity of the plant.**
**Cost per cubic yard of mining, hoisting, and washing.**
**What area of ground is worked from one shaft?**
**If winter dumps are taken out and washed in spring—**
  **Size of dump.**
  **Method of sluicing.**
  **Cost of sluicing.**
  **Is dump rethawed?**
  **Method and cost of rethawing.**
  **Method of sampling dump, if any.**

## ACKNOWLEDGMENTS.

Acknowledgments for assistance rendered and courtesies extended to the party would, if made in detail, fill a volume of themselves. I especially extend thanks to the operators of the Atlin and Klondike districts, and to the officers of the British Columbia and Yukon Territory administration for their hospitable reception to a foreign expedition. To the citizens of Alaska, in whatever capacity they may be employed, the party is indebted for uniform courtesy and assistance as well as unstinted hospitality. Without the cooperation of the miners the work attempted would have been impossible. I also wish to express my thanks to members of the Geological Survey who have furnished me with the valuable results of their Alaska investigations.

I wish to acknowledge indebtedness for assistance in preparing the description of ditch construction in Seward Peninsula to Messrs. G. A. R. Lewington, J. W. Davidson, A. H. More, and William West, of Nome.

Thanks are due to Mr. J. P. Hutchins, a gold-dredging engineer of long experience, who has contributed to the report an authoritative treatise on that subject.

The writer is indebted to Mr. Henry C. Perkins, of Washington, for many suggestions made during the final preparation of the manuscript, especially of that portion dealing with hydraulic mining.

## GENERAL STATEMENT OF CONDITIONS.

Placer mining is that form of mining in which the surficial detritus is washed for gold or other valuable minerals. When water under pressure is employed to break down the gravel, the term *hydraulic mining* is generally employed. There are deposits of detrital material containing gold which lie too deep to be profitably extracted by surface mining, and which must be worked by drifting beneath the overlying barren material. To the operations necessary to extract such auriferous material the term *drift mining* is applied.

As nearly all mining in alluvial deposits comes under the head of gravel mining, that term has been adopted in the main for operations described *in the report* of which the following chapter is a summary.

Occasionally, however, the precious mineral sought lies i of fine sand, or even entirely in the crevices of the bed rock the alluvial deposit rests. Obviously the term gravel m not cover the cases in which detrital gold is extracted matrices, and the general term placer mining[a] has been, added in the title of this report for want of a name w include all operations considered.[b] When in the subsequ the terms gravel deposit, gravel washing, and gravel mining are employed they must be understood, for the sake of brevity, to include the consideration of all classes of deposits in which gold of detrital origin is found.

The term "*elluvial*" has been applied to placer deposits formed by the rotting of rock in place to greater or less depth.[c] Such deposits do not occur in the portions of Alaska visited, and may be excluded from consideration.

In regard to the valuable contents of the deposits, it should be stated that in all the cases here considered gold is the mineral sought. Platinum or any minerals of the platinum group have not been found in paying quantity in any part of Alaska. Alluvial tin has been found and mined in the western portion of Seward Peninsula. The deposits were not, however, visited by the present expedition.[d]

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[a] *Placer*, according to a Spanish definition, is a place near the bank of a river where gold dust is found. Lindley on Mines, sec. 419, makes the following comments:

"Dr. R. W. Raymond (Glossary of mining and metallurgical terms, Trans. A. I. M. E., vol. 9, p. 164) defines the word placer as a deposit of valuable mineral found in particles in *alluvium* or diluvium, or beds of streams. He adds to the definition the statement that, by the United States Statutes, all deposits not classed as veins or rock in place are considered *placers*. As was said by the Supreme Court of the United States (Reynolds *v.* Iron S. M. Co., 116 U. S., 687-695; 6 Sup. Ct. Rep., 601), in distinguishing the two classes of deposits: 'Placer mines, though said by the statutes to include all other deposits of mineral matter, are those in which this mineral is generally found in the softer material which cover the earth's surface, and not among the rocks beneath.'"

It is evident that the term placer mining as used in the present report covers a much more limited field than would be the case were the term placer used in its broad legal sense.

[b] The term *alluvial mining*, used in Australia, is not generally employed in the United States.

[c] Eng. and Min. Jour., vol. 77, May 5, 1904, p. 722.

[d] See the report of A. J. Collier (Bull. U. S. Geol. Survey No. 229, 1904) for an exhaustive account of the York tin deposits.

## CLASSIFICATION OF ALLUVIAL GOLD DEPOSITS IN ALASKA.

The alluvial gold deposits of Alaska may be classified as follows:

*Classification of alluvial deposits in Alaska.*

| | |
|---|---|
| Creek placers | Placers in, adjacent to, and at the level of small streams. |
| Hillside placers | Placers on slopes, intermediate between creek and bench claims. |
| Bench placers | Placers in ancient stream deposits from 50 to 300 feet above present streams. |
| River-bar placers | Placers on gravel flats in or adjacent to the beds of large streams. |
| Gravel-plain (tundra) placers. | Placers in the coastal plain of Seward Peninsula. |
| Sea-beach placers | Placers adjacent to the seashore to which the waves have access. |
| Lake-bed placers | Placers accumulated in the beds of present or ancient lakes; generally formed by landslides or glacial damming. |

The methods of working the alluvial gold deposits are shown in the following table:

*Methods of working alluvial gold deposits of Alaska.*

| Class of placers. | Method of working. |
|---|---|
| Creek placers | Hydraulicking.<br>Hydraulicking with hydraulic elevator.<br>Dredging.<br>Open cutting, separate[a] stripping, and shoveling in.[b]<br>Open cutting, separate stripping, and horse scraping.<br>Open cutting, separate stripping, and steam scraping.<br>Open cutting, separate stripping, wheeling, and cable tram.<br>Open cutting, separate stripping, and steam shovel.<br>Open cutting, separate stripping, track and incline system.<br>Open cutting, separate stripping, track system, with hydraulic elevator.<br>Open cutting, separate stripping, track system, with derricking.<br>Open cutting, separate stripping, skidding, with derricking.<br>Shaft, drifting, and timbering.<br>Shaft, thawing, and drifting. |
| Hillside placers[c] | By the same methods as creek claims. |

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[a] Charged to independent account. The stripping generally consists of frozen "muck," a mixture of silt and ice, which is ground-sluiced off.

[b] Includes rocker work.

[c] Detritus varying from 3 to 60 feet in thickness.

*Methods of working alluvial gold deposits of Alaska*—Continued.

| Class of placers. | Method of working. |
|---|---|
| Bench placers[a] | Hydraulicking.<br>Open cutting, separate stripping, and shoveling in.<br>Open cutting, separate stripping, and horse scraping.<br>Shaft or adit, drifting, and timbering.<br>Shaft or adit, thawing, drifting, little timbering. |
| River-bar placers[b] | Hydraulicking with hydraulic elevators.<br>Dredging.<br>Open cutting, steam shovel. |
| Gravel-plain (tundra) placers.[c] | Hydraulicking with hydraulic elevating.<br>Open cutting, separate stripping, and shoveling in.<br>Shaft, thawing, and drifting. |
| Sea-beach placers[d] | Digging shallow pits and shoveling in.[e]<br>Dredging.<br>Special devices. |
| Lake-bed placers | Hydraulicking. |

[a] Detritus varying from 5 to 150 feet in thickness; in parts of Seward Peninsula to 230 feet.
[b] Detritus from 3 to 60 feet in thickness.
[c] Detritus from 15 to 150 feet in thickness.
[d] Detritus from 1 to 6 feet in thickness.
[e] The greater part of the gold from the beach sands has been obtained by rockers.

The above classes are based on operations actually seen during the season of 1904. Suggestions concerning the application of other methods to certain forms of deposits are given in the body of the main report.

In the districts (Pl. II, p. 14) visited the deposits under exploitation as above classified were as follows:

*Classes of deposits worked in districts visited.*

| Province. | District. | Class of placer worked. |
|---|---|---|
| South Coast | Juneau | Creek and lake-bed placers. |
| Interior | Atlin | Creek and bench deposits. |
| | Klondike | Creek, hillside, and bench placers. |
| | Eagle | Creek and bench placers. |
| | Birch Creek | Creek, hillside, and river-bar placers. |
| | Fairbanks | Creek placers. |

*Classes of deposits worked in districts visited*—Continued.

| Province. | District. | Class of placer worked. |
|---|---|---|
| Seward Peninsula | Nome | Creek, hillside, bench, gravel plain, and seabeach placers. |
| | Council | Creek, hillside, and river bar placers. |
| | Solomon | Creek and river bar placers. |

In the districts not visited the classes of deposits are as follows:

*Classes of deposits worked in districts not visited.*

| Province. | District. | Class of deposit worked. |
|---|---|---|
| South Coast | Porcupine | Creek and bench placers. |
| | Nizina | Do. |
| | Chisna | Creek placers. |
| | Sunrise | Do. |
| Interior | Fortymile | Creek and bench placers. |
| | Rampart | Do. |
| Seward Peninsula | Topkok | Creek, gravel plain, and seabeach placers. |
| | Port Clarence | Creek and bench placers. |
| | Fairhaven | Do. |
| | Kougarok | Do. |

## METHODS OF MINING, AND CONDITIONS.

The mining of placer gold in Alaska is carried on for the most part during June, July, August, and September. The gold-bearing gravel mined during the remainder of the year by winter drifting does not exceed 15 per cent of the total annual amount extracted. The gold can not be washed from this gravel until the cessation of winter conditions liberates the water in spring for sluicing purposes. The sluicing of the "winter dumps" takes place during the latter part of May.

Many of the methods of mining have been developed within the last ten years to suit the unusual conditions existing in the northern gold fields. Gravel miners from other parts of the world found that in Alaska much of their previous experience proved of no special benefit. On the other hand, men without previous experience in mining, but possessing ingenuity, have occasionally adopted devices which have proved efficient and adequate to meet the northern conditions. Methods which had been condemned or tried with ill success in other countries have given good results in Alaska, while the attempts to apply hydraulic or mechanical methods of established reputation elsewhere have frequently resulted in ignominious failure.

Mining operations have been made difficult by the short available season, the lack of grade to the streams, poor water supply, poverty of timber resources, high cost of labor and transportation, concentration of gold on and in the bed rock, and comparatively large thickness of barren overburden, the frozen, or worse still, half-frozen condition of the gravel, lack of wagon roads, and inadequate mining and police regulations. In spite of these obstacles the wide and fairly uniform distribution of alluvial gold over large areas of Alaska, hitherto unexploited, the uniformly healthy and even enjoyable climate of the country, and the near proximity of the phenomenally rich gold fields of the British Yukon Territory, offer a certain justification for the present energetic prospecting and mining for gold over so extensive an area.

The main impressions derived from an inspection of the placer gold fields of the north are as follows: (1) Operations requiring the installation of expensive plants are frequently undertaken before adequate sampling of the ground has been done; (2) the methods of mining and conveying the auriferous material, while often leaving much to be desired from the standpoint of economy, are, in the main, developing along favorable lines; (3) the gold-washing and gold-saving appliances in use are, in numerous cases, inexcusably crude and inefficient.

The winning of gold from alluvial material is a business difficult both to learn and to conduct successfully. The careful miner, like the careful manufacturer, will give as much attention to one part of his business as to another, irrespective of the scale on which it is conducted. The extensive but not remarkably rich gold-bearing area of Alaska offers a field for men who propose to conduct their operations with energy, intelligence, and economy. To others it can afford only ultimate poverty and despair.

The South Coast province is characterized by heavy grades, abundant water supply, and good timber. Gold-bearing gravels are, however, distributed in small quantity and, however good the conditions for the installation of hydraulic plants, the province remains an unimportant producer of alluvial gold.

The Interior province promises to continue for many years a fairly important producer. Geographically considered, the phenomenal Canadian deposits of the Klondike come under this province. No gravels approaching the Klondike deposits in richness have been found on the American side, but a large area yet remains to be prospected.

Owing to the topographic conditions, low grades to creeks, and insufficient water supply at an available elevation, hydraulicking on any but the smallest scale is impossible. Many of the creek deposits are shallow, and, besides the primitive method of shoveling into sluice boxes so largely in practice, there is a considerable field for the installation of horse-scraping methods and the installation of simple mechanical plants. Solidly frozen creek deposits exceeding 15 feet in depth can be most economically worked by drifting methods, as heretofore. Experience gained in the Klondike has been invaluable to the miners now developing the new Fairbanks field. There is room, however, for considerable improvement and reduction of expense in the methods employed.

The natural conditions prevailing in the Alaska interior gold field imply great age and erosion subsequent to any deposition beneath sea level. Topographic conditions exercise a remarkable control over the methods which can profitably be employed in gravel mining, and the prospective miner neglects a vital part of his preparation if he does not study the topographic features of a given district in detail before entering upon his operations. In California and Australia the geologic and topographic conditions favor the placer miner. In other countries, notably Siberia, Alaska, and the Yukon Territory, they are inimical to his success. In Alaska, as a rule, alluvial gold is almost entirely lacking where timber and water are plentiful, grades are steep, and the ground is unfrozen. Where gold is distributed in paying quantity, water supply is inadequate, timber is poor or altogether lacking, the miner must provide grade for his boxes and dump for his tailings by artificial means, and must meet the formidable obstacle of solidly frozen alluvium. Bench deposits, where gravel can be moved on natural grade, occur in both the Fortymile and Rampart districts of interior Alaska, and have been made to produce a small amount of gold by the hydraulic method. Although it is not impossible that extensive and valuable bench deposits may yet be found, no deposit comparing either in extent or in richness with the famous "White Channel" of the Klondike has been discovered.

In that portion of the Alaska interior lying between Circle, on the Yukon, and Fairbanks, on the Tanana, the mountains rise to heights of from 1,500 to 2,000 feet above the level of the streams, have rounded tops, and slope to the intervening valleys at angles which do not exceed 20 degrees, and often are not greater than 10 degrees. The streams and valleys are on a gently descending plain, the grade of which does not exceed 3 per cent, except in the upper one-half mile, and frequently is not over 1 per cent. The mountains are referred to by the inhabitants as "domes," and the word fairly well describes them. They present what corresponds most nearly to the upper segment of a great ellipsoid except in places where the erosion has not been sufficient to accomplish the obliteration of a still more ancient topography. This ancient surface, remnants of which are visible on the tops of the highest mountains, was evidently a base-leveled plain, which was approximately 2,500 feet above the present drainage plain. Although the base-leveling is apparent to the eye, it is not evinced by the presence of rounded gravel on its surface. The lack of gravel is accounted for by the fact that the second denudation has progressed for a great period, and the comparatively small amount or vertical section of gravel which existed subsequent to the elevation has been worn away.

In the Klondike recent streams have cut the old Pleistocene channels and have reconcentrated the gold.[a] The gold is about equally distributed in the old and in the new gravels. From the miner's standpoint, therefore, in the Klondike region there are two great classes of mining to be considered, namely, creek mining and bench mining. Outside of these two classes there is no mining in the Klondike of productive importance.

In the Birch Creek district, especially on Deadwood Creek, there is a very small amount of gravel in low benches which may be termed hillside deposits. The bulk of the mining, perhaps 90 per cent of it, is creek mining in its various forms. The terms bench deposit, hillside deposit, and the like are very loosely applied by the miners of the Northwest, and names are given to classes of mining to which they do not in any sense apply. This looseness of nomenclature is apparent in the Fairbanks district, where the term bench mining is applied on Cleary Creek to operations in progress on the left bank of the stream at a place one-fourth mile above the junction of Cleary and Chatham creeks. But whereas the depth to bed rock in the main creek at this point is 18 feet, the depth on the so-called bench, 700 feet to the left, is 53 feet, and the level of the bed rock at which the gravel is found is practically the same. In the one case, namely, in the creek working, the overburden is 6 feet of muck, while in the "bench" to the left the overburden is over 45 feet of muck. The gently sloping side of the valley at this point is unbroken in outline.

Observations along the various producing creeks and from the hilltops have failed to distinguish any traces of bench topography in the Fairbanks district. Such placer mining as is carried on there comes under the head of creek mining. Geological evidence, however, suggests that bench deposits occur in the region lying between the Fairbanks and Rampart districts.

The methods applicable to bench mining at Dawson can not be used in the Fairbanks district, and all thought of applying them must be eliminated. The country being in every sense of more gentle topography, there is no room for the disposal of tailings from bench operations conducted by hydraulicking.

On Pedro and Twin creeks there are about 2 miles of ground less than 15 feet in depth which can be worked by open cutting, either by shoveling into sluice boxes or by derricking. On a portion of this ground it is possible to handle the water by bed-rock drain. Open-cut mining has also been successful on Chatham Creek near its junction with Cleary. In all other portions of the district, so far as developed, drift mining according to the Klondike system of thawing either by steam or hot-water hydraulicking, hoisting, and conveying by means of the self-dumping bucket on cable tram will probably be found most economical. I would suggest the method of underground hot-water hydraulicking to the miners of Cleary Creek, while on Fairbanks Creek steam thawing appears to be advisable. The efficiency of the hot-water method as used in the Klondike is from 5 to 6 cubic yards per horsepower generated in the boiler as against 3 cubic yards with steam. The method can, however, be applied only under certain favorable conditions.

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[a] *See McConnell, R. G.*, Preliminary report on the Klondike gold fields: Geol. Survey Canada. 1900.

In Seward Peninsula the greater rainfall, larger catchment areas at the heads of the long rivers, and the comparative cheapness with which ditches can be constructed has led to the investment of much capital in long water conduits. For example, a ditch system of 54 miles, built at an expenditure of $300,000 and costing $15,000 annually to maintain, supplies 2,000 miner's inches of water at 360 feet head for four months in the year. Approximately 200 miles of ditches have been built in various parts of the peninsula. Excavations of earthwork for ditch building in the peninsula average $1 per cubic yard.

Hydraulicking without the use of hydraulic lifts is economically impossible, except in extremely rare cases. Bench gravels in the front of Anvil Mountain, facing the sea, can be hydraulicked if water at a sufficient head can be obtained at an expense which is not prohibitive. The remarkable ancient gravel channel which cuts the southern portion of the peninsula from east to west, extending from Fish River along Casadepaga and Kuzitrin rivers to Port Clarence, lies at so low a level that the present streams have not cut through it to bed rock. Except where subordinate pay streaks exist in it above the present stream, therefore, the physiographic conditions will forbid its gravels being hydraulicked, while any other form of open cutting is manifestly impossible. It has been little explored, and portions of it may be found rich enough to drift.

Horse scraping, steam or power scraping, derricking, and the application of the mechanical shovel, accompanied in most cases by ground sluicing of the frozen muck, should receive consideration from the creek operators in Seward Peninsula, where the deposits are less than 15 feet in depth. The low price of winter labor ($2.50 a day and board) should permit of an increasing amount of winter drifting work throughout the peninsula.

It will doubtless eventually be found that the power of water under pressure can be more successfully applied to the working of the average Seward Peninsula placer by generating electric power and applying it to various mechanical devices. While it can not be denied that some of the hydraulic elevator installations are handling the gravel at a profit, the contrivance is a makeshift, and its use forms no part of bona fide hydraulic mining.

### COST OF MINING.

The average value of fuels in Alaska as shown by present operations is as follows:

*Cost of fuels available for use in Alaska.*

| | |
|---|---|
| Bituminous coal, price at Nome | $17 per ton (2,000 pounds) |
| Crude oil at Nome | $3 per barrel |
| Spruce wood, average price in the interior | $12 per cord |

Experience in the Nome district indicates that California crude oil is the most economical fuel available in the southern part of Seward Peninsula. In the interior of Alaska the price of imported crude oil renders its use prohibitive for mining operations.

The recently exhibited tests of the adaptability and efficiency of gas-producing engines should receive attention from operators who contemplate the installation of mechanical plants in any part of Alaska. There is no question that bituminous coal and lignite can be utilized for gas producers, giving proportionately better results than anthracite. An engineer operating a large pumping plant in the Klondike is of the opinion that even the poor spruce wood available for fuel in interior Alaska can be utilized in the gas-producer engine. The prejudices which exist againt the explosion type of engine in the United States are fast disappearing. They have been due to faulty construction of the engines and lack of knowledge of their principle among those who attempt to operate them. The present valid objections to installing gas and gas-producer engines are that these engines are undergoing a process of evolution, and the standard has not been attained. According to Mr. M. R. Campbell, the Government coal-testing plant at St. Louis has demonstrated that a gain of from 30 to 50 per cent of efficiency is attainable in the gas-producing as compared with the steam-producing engine.

The comparatively low cost of California crude oil at Nome renders it a valuable fuel for the mining operations in that vicinity. The satisfactory results from one type of gas engine at St. Louis showed that crude oils of widely varying composition can be used with a higher efficiency in generating gas for explosive engines than in generating steam.

The purchase of water for hydraulic or sluice purposes is not general in Alaska. In Seward Peninsula water under natural head or pumped water is sold to miners to a limited extent. The average price is $1 per miner's inch, twenty-four hours' service, for water under head and 50 cents for sluice water. The inch used corresponds to 1.2 cubic feet per minute. This measure of the miner's inch is *not accepted in* this report. The miner's inch, according to its best definition, which is followed in this report, corresponds for all practical purposes to a flow of 1.5 cubic feet per minute. It is to be hoped that if the Federal Government ever succeeds in establishing an adequate code of mining law for its possessions a definition of the miner's inch will be included.

The data in the table on page 38 were compiled from statistics collected during an inspection made in the summer of 1904 of the placer fields in Alaska, Yukon Territory, and northern British Columbia. Of the statements furnished by operators, only those which are considered reliable have been used. The work attempted had no relation to the sampling or valuing of mining properties, and time did not permit, except in a few cases, the measuring of the ground.

Owing to the varying conditions governing the cost of mining in the North, the Territory has been divided into three provinces. The South Coast province includes the Juneau, Porcupine, and Sunrise districts of Alaska. The Interior province includes the Atlin district of British Columbia, the Klondike district of Yukon Territory, and the Fortymile, Eagle, Birch Creek, Fairbanks, and Rampart districts of Alaska. The Seward Peninsula province includes the Nome, Council and Solomon districts of Alaska.

The Nizina district of the South Coast province and the Port Clarence, Fairhaven, and Kougarok districts of Seward Peninsula, none of which were visited, are separately considered.

In preparing the sheet the working costs of 118 different operations were first tabulated with reference to the method employed and to situation. A second table was then prepared, in which the working cost was augmented by an amount per cubic yard based on allowance for depreciation of plant. In general, six years was taken as the average life of an individual property, and, except in the case of winter drifting operations, one hundred and twenty days as the working season. It was then assumed that five annual payments are made to a depreciation fund. The fund is equivalent to the cost of plant and maintenance during the life of the property plus six years' simple interest on the investment at 5 per cent. Each annual payment was divided by the season's output in cubic yards, and the amount thus obtained was added to the daily working expenses, to get the total output cost per yard, as far as possible. Prices paid for mining property are not considered, as they represent an unknown factor.

In cases where expensive plants have been installed the amortization was calculated separately for each case.

In cases of shoveling-in and small mechanical plants, the installation and maintenance cost was taken at an average amount for a group of operations in each district. Where the operation implies an additional stripping of overburden, which is always separately charged, the cost was distributed and added to the gravel extraction cost.

From the second table, where the costs were reduced to one figure for each district, a third (table 1, p. 38) was prepared, giving as nearly as possible the average cost for each of the seventeen separate methods considered in one or more of the three provinces. Where the operations from which the averages are derived exceed two in number, the fact is so indicated in the table.

The attempt has been made to reject figures which were evidently not representative. The final figure arrived at is not, however, always satisfactory. For example, under No. 5 (the method of working open cut by shoveling into wheelbarrows, wheeling to bucket, hoisting and conveying to sluice by self-dumping carrier on cable) $2.14 is representative for the Klondike, where seepage water is generally pumped from the pit, and many operators pump the water for sluicing. On the other hand, at a plant in the Birch Creek district of Alaska, mining only 22 cubic yards per day and handling the water by a drain, the cost of operation was $1.50 per cubic yard. In No. 13 (drifting solidly frozen ground, steam or hot-water thawing, hoisting and conveying with the use of the self-dumping bucket) the cost in the Klondike is $1.95, while the higher figure given is arrived at by combining the expensive American camps of Fortymile and Fairbanks, where the cost is $4.63 and $3.56, respectively.

The high cost of hydraulicking with use of hydraulic lift in Seward Peninsula is caused by the difficulty of moving the gravel to the bed-rock sluice[a] and the expense of the ditches and installations. Hydraulicking by means of water under natural head without the use of the hydraulic lift, or some other means of elevating the material, was not seen by me in Seward Peninsula. It is understood that a hydraulic plant is in successful operation at Bluff, 50 miles to the east of Nome, but there are no data at hand concerning it.

In the interior only bench gravels are hydraulicked. Steeper grades for sluices can be obtained, and the gravel is more easily moved. The high duty of the miner's inch in the Klondike is a large factor in bringing down the cost of No. 1 and No. 16.

It should be distinctly understood, if hydraulicking costs in the interior appear attractively low, that the water supply is exceedingly variable, and that no reliable estimate can be made beforehand of the output of a given season's operations. Furthermore, while much of the bench gravel was originally rich, the pay streaks have been largely drifted out, and the gold is not disseminated through the upper portion of the gravel to the extent that it is in the California gravel banks. With regard to the pumping of water for hydraulicking purposes, the practice can not be too strongly condemned. He is a bold man who attempts it, and a singularly fortunate one who makes a financial success of it.

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[a] *This is caused not* only by the exceedingly gentle grades of the streams, but also by the shingly *character of the material* handled.

Mr. Stephen Birch, operating in the Nizina district of Alaska, has courteously furnished for this report a summary of the costs of working placer ground on Dan Creek. These figures are given separately (p. 39) after the table, as they imply a total charge of invested capital in addition to working costs against one season's operations. They are worth the attention of prospective placer miners.

The cost of shoveling into sluice boxes in the remote parts of Seward Peninsula is at present from $3 to $5 per cubic yard. Some drifting operations have been carried on in the Kougarok and Fairhaven districts, on which figures are not at hand.

Dredging estimates furnished by reliable interior operators place the cost at 80 cents per cubic yard where gravel must be thawed by points ahead of the dredge. In Seward Peninsula it is estimated that if the property is sufficiently large for a ten-year life, a dredge can be operated at the cost of 30 cents per cubic yard. The field for dredges in placer mining in Alaska is extremely limited. In Seward Peninsula it is not impossible that some of the wide, shallow creek deposits will be worked successfully by means of the steam scraper. The cost of an experimental operation on Ophir Creek was reported to be under 20 cents per cubic yard.

The costs of operating by two mechanical systems in Seward Peninsula (involving the labor of men in shoveling into cars and tramming to the bottom of an incline, or into a bed-rock sluice leading to hydraulic elevator throat) are unfortunately not available for publication. The derricking system, No. 7, however, both in the interior and in Seward Peninsula, appears to be superior in point of cost to either of the above mentioned methods for working the average Alaska open cuts.

Frozen ground can not be attacked with success by the steam shovel. Even where it digs the gravel successfully, if men follow it to clean bed rock by hand, the cost of operating is sometimes doubled. The steam shovel has, however, a field in northern placer mining.

Regarding mechanical operations in general, the important principle should be emphasized that the main expense is getting the material into the receptacle which conveys it to the sluice or washing plant. Tramming even for a long distance, and to a considerable elevation, adds a very small proportionate amount to the total cost of working. The establishment of a permanent washing plant, economically situated as regards water supply and dump, should be considered by every Alaskan miner who proposes working the shallow creek deposits which characterize that country. The isolation of the washing operations, together with the adoption of the most economical system of tramming possible, will go far toward attaining the ends of adequate grade and room for tailings, which are the essential features of successful gravel mining.

TABLE 1.—*Average capacity and cost of gold-gravel mining operations in northwestern America.*[a]

| | 1. Hydraulicking, no pumping of water. | 2. Hydraulicking with use of hydraulic elevator. | 3. Open cut; shoveling into sluice boxes, including stripping top dirt; no pumping. | 4. Open cut; horse scraping. | 5. Open cut; shoveling; wheeling to bucket; cable tram to sluice. | 6. Open cut; shoveling into cars; track and incline to sluice. | 7. Open cut; shoveling into buckets or skips; skidding or tramming, and derricking to sluice. | 8. Open cut; shoveling into sluice; tailings by hydraulic lift. | 9. Open cut; steam-shovel excavating; track and incline to sluice. | 10. Open cut; steam scraping; generally on stripping work or tailings. | 11. Dredging. | 12. Drifting partly frozen or thawed ground requiring timbering. | 13. Drifting and thawing solidly frozen ground; little or no timbering. | 14. Winter drifting and spring sluicing of dumps. | 15. Mining or stripping overburden by ground sluicing. | 16. Hydraulicking by means of pumped water. | 17. Booming with self-dumping water gate. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| SOUTH COAST PROVINCE. | | | | | | | | | | | | | | | | | |
| Number of operations considered | 6 | 6 | 6 | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ |
| Capacity, cubic yards, in 24 hours | 833 | 350 | 54 | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ |
| Thickness of deposit, feet | 30.3 | 25 | 5.6 | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ |
| Thickness of gravel worked, feet | 30.3 | 25 | 3.7 | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ |
| Cost[b] | $0.20 | $0.31 | $2.01 | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ |
| INTERIOR PROVINCE. | | | | | | | | | | | | | | | | | |
| Number of operations considered | 13 | ........ | 20 | ........ | 8 | ........ | ........ | ........ | ........ | 6 | ........ | ........ | 7 | ........ | ........ | 4 | ........ |
| Capacity, cubic yards, in 24 hours | 1,049 | ........ | 63 | 105 | 162 | 450 | 233 | 184 | 800 | 92 | 1,062 | 50 | 75 | 50 | 150 | 830 | 250 |
| Thickness of deposit, feet | 37.4 | ........ | 8.6 | 20 | 17.5 | 14 | 15 | 8 | 22 | 15 | 35 | 60 | 26.4 | 26.4 | 9 | 33 | 7.5 |
| Thickness of gravel worked, feet | 37.4 | ........ | 3.5 | [c]10 | 4.5 | 5 | 9 | 6 | 22 | 8.7 | 35 | 4 | 4.36 | 4.36 | [d]9 | 33 | [c]6 |
| Cost[b] | $0.238 | ........ | $2.39 | $0.60 | $2.14 | $2.43 | $1.75 | $1.25 | $1.46 | $0.49 | $0.49 | $4.25 | $3.38 | $5.14 | $0.17 | $0.65 | $0.07 |
| SEWARD PENINSULA PROVINCE. | | | | | | | | | | | | | | | | | |
| Number of operations considered | ........ | 4 | 10 | 5 | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | 3 | ........ |
| Capacity, cubic yards, in 24 hours | ........ | 658 | 145 | 200 | ........ | ........ | 550 | ........ | 1,000 | ........ | 700 | 80 | 20 | 83 | 173 | 250 | ........ |
| Thickness of deposit, feet | ........ | 12 | 6.6 | 5 | ........ | ........ | 15 | ........ | 30 | ........ | 8 | 20 | 35 | 85 | 4 | 23 | ........ |
| Thickness of gravel worked, feet | ........ | 12 | 3.3 | 5 | ........ | ........ | 11 | ........ | 27 | ........ | 8 | 7 | 4 | 4.3 | [d]4 | 23 | ........ |
| Cost[b] | ........ | $0.89 | $1.87 | $0.46 | ........ | ........ | $0.91 | ........ | $0.52 | ........ | $0.43 | $4.49 | $3.66 | $4.61 | $0.10 | $0.93 | ........ |

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[a] Lost time, the prices paid for mining property, and the cost of equipment other than that relating to actual mining (e. g., railways, wagon roads, etc.) are not taken into account, and any estimates based on these figures must make due allowance for these expenses; otherwise the costs here given will be found two low.

[b] Dollars per cubic yard.

[c] "Muck" and top gravel.

[d] "Muck" or fine silt and ice; from 50 per cent to 75 per cent ice.

Mr. Stephen Birch, in a letter, gives the cost of placer work on Dan Creek, Nizina district, Alaska, as follows:

By ground sluicing through 20-inch flume: 6,803 cubic yards, $8,781.44, or $1.143 per cubic yard.

By use of 8-inch cotton pressure-hose and nozzle through 20-inch flume: 1,600 cubic yards, $1,457, or $0.91 per cubic yard.

By use of pick and shovel only, through 10-inch sluice box: 2,320 cubic yards, $5,100, or $1.875 per cubic yard.

A 273-foot tunnel, 6 feet by 6 feet, timbered: $1,017, or $3.725 per running foot, or 407 cubic yards of gravel removed, which costs $2.50 per cubic yard.

While the cost given above may seem high, it is because of the fact that it includes the cost of the tools and material now on hand, which were necessary to remove this gravel. Now, if this work is continued on for a number of years, the depreciation of the tools, etc., could by charged proportionately. These prices may not be a criterion for future operations in that country but were our first cost of operation, and any strangers going into that section of country would be apt to run up their costs to these figures.

## PROSPECTING.

Creek claims must be prospected by shafts, open cuts, or drill holes. Shafts are generally used and are most practicable for the Alaska miner. Open cuts are not practicable, except in very shallow ground. Drilling requires expensive equipment, the machines costing from $3,500 to $4,000 delivered in Alaska. As a means of prospecting auriferous gravels drilling is very satisfactory, but it is generally employed only in deep gravel beds, where dredging is contemplated.[a] In such ground excess of water prohibits the sinking of shafts. The cost of drilling with a 6-inch core churn drill is $2.50 per foot in California, and the tests of the deep gravels of the coastal plain have shown it to be practically the same in the vicinity of Nome, on Seward Peninsula. The cost of drilling depends on the price of labor and fuel, and will probably be about $3.50 per foot in the interior of Alaska. Unless contractors are found already equipped to do the work, drilling is more expensive than sinking pits, the cost of which varies from $3 to $8 in Alaska placer fields.

In the solidly frozen ground encountered in northern placers it is not necessary to sink the casing around the pipe as in ordinary drilling. In partly frozen ground, where the casing is necessary, much difficulty has been found in pulling it up. A driller of the churn type is shown in Pl. III, *A* (p. 40), mounted on a scow for prospecting the bed of Snake River, Seward Peninsula.[b] The results obtained by drilling when compared with actual tenor of the ground are variable. In testing the bed of Solomon River the results were found to run 7 per cent lower. In all parts of Alaska drill holes should be not less than 100 feet apart, on account of the unequal distribution of the gold. Geologic conditions at Oroville, or in other dredging fields of flood-plain character, are radically different. There the ratio of holes is 1 to an acre or 1 to 5 acres.

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[a] See a further description of prospecting by drill holes, by Mr. J. P. Hutchins, in the chapter on dredging, later in this report.

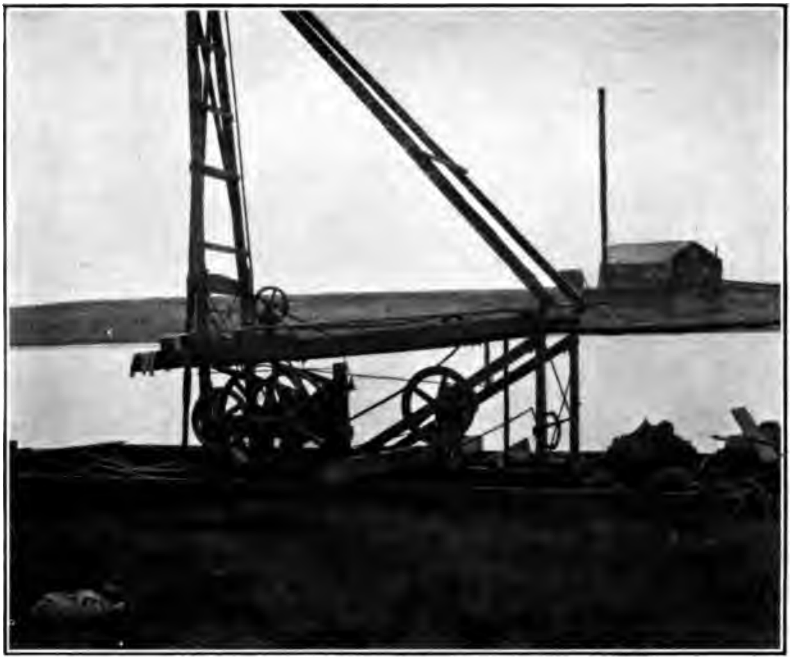
[b] For description of drilling operations in testing placer ground see Knox, N. B., Dredging and valuing dredging ground at Oroville, California: Trans. Inst. Mining and Metallurgy, June 18, 1903. Also Smith and Stebbins, Gold dredging at Oroville: Eng. and Min. Jour., Dec. 8, 1904.

Shallow bench gravels can occasionally be prospected by diverting a high stream or water from a ditch in a direction transverse to the gold-bearing channel. The water will ground sluice a trench to bed rock, thus crosscutting the ground. Such prospecting is done in the Fortymile district, and an example of it was seen on Gold Bottom Creek, in the Council district of Seward Peninsula.

Bench gravels covered by heavy overburden are usually prospected by drifts. Drifts require timbering and are more expensive than shafts, but give a more satisfactory test of the ground. In rich pay streaks the running of prospect drifts often more than pays the cost. Owners of claims sometimes get their ground partly prospected by letting out the right to drift to two or more men, who pay a royalty on the gold they take out. In the Nizina district a 20 per cent royalty is charged on prospect work.

When ground is prospected by shafts the distance between the shafts varies, although the variation is not so great as when the prospecting is none by drill holes. The practice in Alaska varies according to the work, the resources of the owner, and the depth of the ground. Where the gold has not traveled far and in consequence the values are spotted, shafts should be sunk 30 to 50 feet apart, but such close work is never done. Twenty shafts to a 20-acre claim are common. Shallow ground, 6 to 18 feet deep, can be well and cheaply prospected in the winter months. Deep ground, as in the Fairbanks field, is expensive to prospect, and the prospect shafts do not average four to a claim. The cost of sinking shafts in the various camps is given below.

Twenty shafts, 3½ by 6 feet and 12 feet in depth, timbered in thawed ground, or untimbered in frozen ground, cost from $1,200 to $2,000. If steam thawing is employed, the outfit, consisting of a 6-horsepower boiler, three or four steam points, steam hose, and pipe connections, will cost from $300 to $500. Hot stones, hot water, and wood fires are more generally employed in prospecting ground in the remote camps. The hardest ground to prospect is that which is partly frozen. For example, in the Birch Creek and Fortymile districts shafts 15 to 25 feet in depth penetrate frozen ground and require no timbering until they are within 5 feet of bed rock, when a rush of water is frequently encountered. The entire labor of sinking the shaft through the overlying barren material may be lost, as the shaft may be flooded before pay gravel is reached.



A. CHURN DRILL MOUNTED ON SCOW SNAKE RIVER, ALASKA

B. WINTER DUMPS, KLONDIKE.

## COST OF PROSPECTING IN DIFFERENT LOCALITIES.

In the Juneau district prospect shafts must be square set, and cost on an average $6 a foot, and in heavy landslide $20 a foot. In the Porcupine district the cost of a cut 25 by 12 feet and 40 feet deep is $50 per foot, and that of an 8- by 8-foot shaft is $20 per foot. In the Sunrise and Chisna districts ground is prospected by open cuts. A cut 50 feet in length is reported to cost $2,000.

At Atlin ground is prospected mostly by drifting. On McKee Creek tunnels, timbered and lagged, cost $3.50 per foot. On Spruce Creek the cost of posts and caps, 10 inches thick and 6 feet long, is 50 cents each; lagging, 10 cents each. On Gold Run thirty 6-inch holes are said to have been drilled to a depth of 32 feet by a churn drill (cost, $3,500, laid down), at the rate of $1 a foot.

In the Klondike drifts to prospect the bench gravels cost $7 to $8 a foot, timbered, and shafts from $5 to $10 a foot. In timbering, three sets of posts, sill, cap, and lagging are put in for $6, and as one-half cord of wood is used to a set the whole cost is $7 per set. In frozen creek ground two men, working three shifts, sunk a pit 5 feet square, 28 feet deep, using about 2-horsepower steam during thirty hours.

On Fortymile Creek, thawing ten hours, two men take out on an average 4 feet a day, the shaft being 5 feet by 3½ feet in dimensions. On American Creek prospecting is very difficult, as running water is always encountered at bed rock, even in the coldest weather.

In the Fortymile district shafts averaging 3½ by 6 feet, untimbered, cost from $3 to $5 per foot. Seven shafts 4 by 8 feet and 23 feet deep cost $2,000. A shaft 7 by 3½ feet and 20 feet deep cost $5 per foot, the thawing being effected by means of wood fires and steam.

In the Birch Creek district shafts thawed down by wood fires to a depth not exceeding 20 feet cost $5 per foot. On Mastodon Creek, in this district, 11 pits 3 by 6 feet and 20 feet deep cost $650. Twenty pits 3 by 6 by 13 feet cost $7 per foot. The differences in cost here are due to the varying amount of permanent frost in the ground, prospecting always being cheapest in solidly frozen ground. On Mammoth Creek 100 pits 10 feet deep cost $5 per foot. Wood fires were used, 2 feet being sunk each day. No timbering was required, and labor was paid $10 a day.

In the Fairbanks district the prospect shafts cost from $7 to $10 per foot. Timbering is generally necessary, but very light poles are used either as lagging or cribbing. The best system is to put in 6-foot sets of 3-inch poles, and outside of these to lag with 2-inch poles vertically, filling in solidly between the poles and the muck or gravel with moss. Shafts cribbed with poles horizontally are more likely to get out of plumb. Four shafts sunk on lower Fairbanks Creek cost $5 per foot, and were 32, 44, 53, and 54 feet deep. On Cleary Creek a shaft 4 by 6½ feet and 75 feet deep (hillside claim), cribbed with 3-inch poles, cost

$560. Wood fires were used in thawing. On Pedro Creek 19 pits 3½ by 7 by 12 feet cost $1,140, and on an adjoining claim pits 3 by 6 feet and 18 feet deep cost $6 per foot. The light timbering and moss filling used in the Fairbanks mines is to prevent the muck walls from thawing and caving.

In the Rampart district 30 pits 3 by 6 feet and 20 feet deep cost from $75 to $100 each. The ground was partly thawed, but required no timbering.

In the Nome district, on account of the entire lack of native timber, shafts are cribbed with 1 or 2 inch planking, set close together, no sets being used. On the Snowflake bench claim, on Anvil Creek, the cribbing was reenforced by 2 by 4 inch posts in the corners. Five dollars per foot is the average cost of 4 by 5 shafts, timbered, in unfrozen or partially frozen ground.

On Bonanza Creek, near Nome, pits in the shallow creek bed, 3 by 7 feet and 7 feet deep, are said to cost 50 cents per foot.[a] The ground is only lightly frozen and requires no timbering. On Ophir Creek, in the Council district, it costs on an average $5 to sink pits 7 feet deep.

In the northern part of Seward Peninsula the ground is solidly frozen. In the Kougarok district pits 35 feet in depth, with steam thawing, cost $8 per foot, and require no timbering.

In the Candle or Fairhaven district, adjacent to Kotzebue Sound, 28 pits 12 to 15 feet deep cost $4 a foot. It was necessary to thaw to bed rock. The equipment cost $500.

## RESULTS OF PROSPECTING.

Numerous reports from many parts of Alaska indicate that the amount of gold obtained per cubic yard from prospect shafts does not equal that extracted by subsequent actual mining. The reverse is very rarely true. It is difficult to assign a reason for this discrepancy other than that, owing to the frozen condition of the gravel, some of the gold escapes when small lots are washed in the winter. Frozen gravel does not easily disintegrate, even in hot water, and unless the residue from panning or rocking is saved and rewashed, losses very likely occur.

It is needless to urge the importance of prospecting ground in a thorough manner before expensive machinery is installed. The many failures through the long history of mining which have resulted from precipitate expenditure to exploit supposed valuable properties present an open page of admonition to him who cares to read.

The sampling of winter dumps as they are extracted does not appear to have received the attention which it deserves. A method used by Mr. Kelley, of Dominion Creek, in the Klondike, is as follows:

The ordinary conical dump of frozen gravel, Pl. III, *B* (p. 40), assumes a somewhat steeper angle than that of loose material. One measured in the Klondike had a slope of 40°. From experience it has been found that if the ground is of uniform richness, 80 per cent of the values are contained in the upper two-thirds of the dump, which has a content of approximately 8,000 cubic yards. The apex of the dumps is generally 30 to 40 feet above the base. Four times each day 5 pans are taken in sampling—one from each quadrant of the dump one-half way down from the top—and one pan from the apex. The results of the 20 pannings are put together before weighing, and 50 per cent of the result is taken for the average value of what has been taken out during the day.

---

[a] *This work is probably* done in the winter with labor at $2.50 a day and board.

Neglect to apply some form of sampling to the dumps has caused many lamentable failures in the Klondike. Winter operators of the Fairbanks district should keep themselves assured by constant sampling that the gravel they are getting out at such a cost carries values.

## WATER SUPPLY.

### CLIMATIC CONDITIONS.

Land areas in high northern latitudes are characterized by distribution of their natural resources that is unfavorable to economic use. In Alaska the great forests that characterize the south coast are replaced by stunted growths of spruce in the gold fields of the interior and by willow copses in Seward Peninsula. During a normal year the rainfall at Juneau, as may be seen from table 2 (p. 48) compiled by Dr. Cleveland Abbe, jr., is over eight times as great as at Eagle, on Yukon River. The catchment area at Juneau is only 4 square miles, and, as the grades of the hillsides are precipitous, the rainfall quickly runs off. A small amount of the water is used for power in connection with milling operations and for mining placer gold. The short, steep creeks and rivers of the south coast have small catchment basins, and even were there any use for the water for placer-mining purposes, there would be difficulty in impounding it after the melting of the snows in June and July.

Large catchment basins exist in the interior of Alaska, but the water to fill them is unfortunately lacking. At Eagle, where the annual rainfall is 11.4 inches in a normal year, the drainage area of Mission Creek and its tributaries which flow into the Yukon is nearly 200 square miles. Of the amount of water caught in a given area, however, very little is available for mining purposes. As an illustration take the case of American Creek, a tributary of Mission Creek. The only placer mining on it is done at a point about 12 miles distant from its mouth, at an elevation of 1,600 feet. As the use of water for hydraulic mining requires a head of 200 feet, and as the height of the divides which surround American Creek does not exceed 3,000 feet, it is evident that the only catchment area available is that lying between 1,800 and 3,000 feet, and between the mine and the head of the creek, or its short tributaries. Of a total catchment area of nearly 75 square miles for American Creek, only 15 square miles are available for the operations at the point referred to. If it were possible to catch all the water, say 11.4 inches in depth, which fell on this area during a season, and use it in hydraulic mining, it would amount to 1,500 miner's inches available for one hundred and twenty days, and would move gravel at the rate of 1 cubic yard to the inch of water. As in practice it is not possible to utilize at head over one-third of that theoretically available in the catchment area, the capacity of the water power available is reduced in proportion. (See fig. 1.)

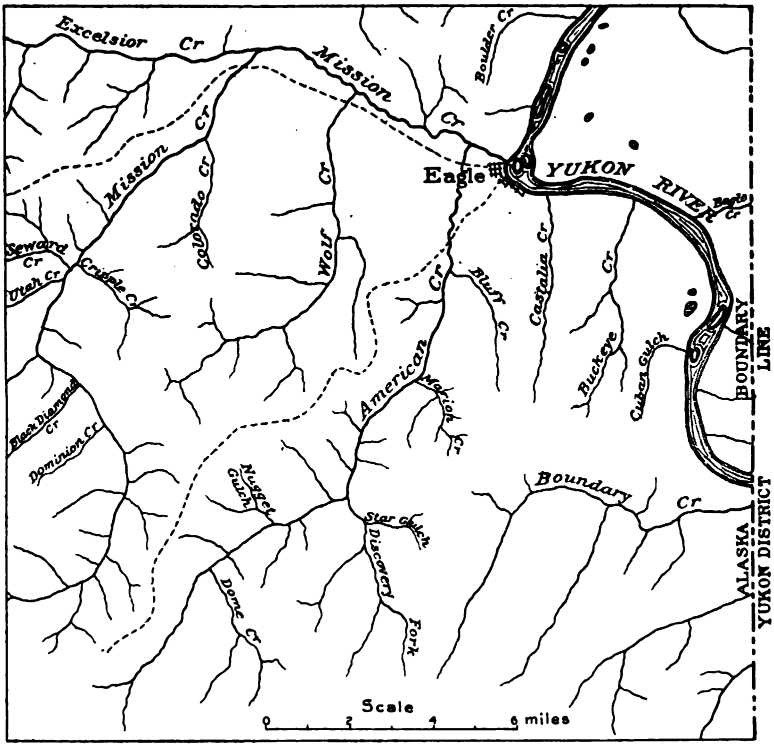


FIG. 1.—Map showing American Creek drainage, Eagle district.

In certain parts of the Fortymile region it is not impossible that water conduits from 10 to 15 miles in length can be constructed to bring water to the placer diggings. The Glacier Mountains, lying west of the upper portion of Mission Creek, rise to heights of 6,000 feet, or 3,000 feet higher than the general level of the interior country, and should afford a more promising water supply than any other area adjacent to the interior placer mining districts. The great drainage system of the South Fork of Fortymile Creek may also be available *to a less extent* for obtaining water under pressure. Miners should bear in mind, however, that even if the water is at hand and the gravel deposit extensive, the attempt to work by the hydraulic method where a grade of 6 inches to 12 feet can not be obtained will probably fail.

In the Fairbanks and Birch Creek districts the rainfall may be assumed to be no greater than that at Eagle. The catchment area that lies at an elevation sufficient to supply the placer ground with water at a proper head for hydraulic operations is insignificant. Even were the auriferous gravels in that district so situated as to be hydraulicked, water for that purpose could not be obtained. The small amount of water which runs in the creek bottoms is available at heads not exceeding 25 feet, by means of ditches, for sluicing purposes. It is of variable quantity, however, and must be husbanded with the greatest care. The promiscuous cutting of timber and stripping of moss, which are the inevitable accompaniments of the exploitation of the placers, will tend to gradually diminish the supply of water. (See figs. 2 and 3.)

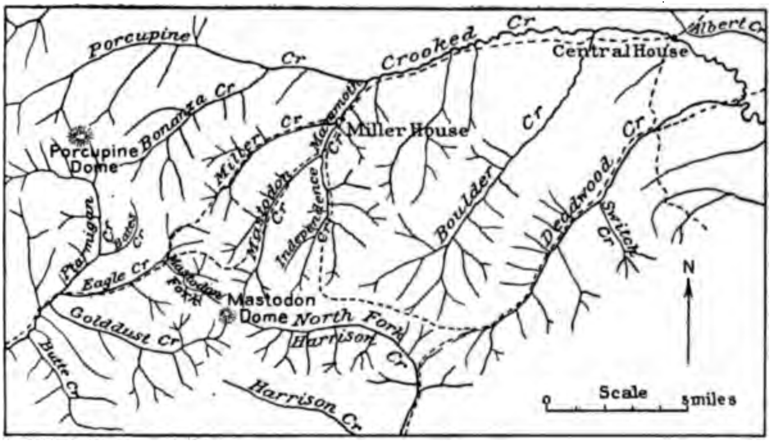


FIG. 2.—Map of part of Birch Creek district.

Of the interior districts the Rampart and Fortymile appear to be most copiously supplied with water, and these are the only districts in the hitherto developed interior where considerable bench deposits have been found. It seems not improbable that a survey with reference to locating the larger catchment areas at sufficient elevation may result in obtaining water for hydraulicking the bench gravels.

No data are at hand with reference to the rainfall in Seward Peninsula. It may be assumed to be larger than that of the interior, and is probably fairly represented by the figure given for St. Michael, 18.1 inches, which fell during the summer of 1904. This amount, however, was much less than that of the year before, and the gold product of the peninsula was thereby materially reduced. The facilities for making use of the water for mining purposes are better on

Seward Peninsula than in the interior. The available catchment areas on the peninsula are larger, and the construction of water conduits, owing to peculiar conditions which will be described, can be undertaken more cheaply. The Kigluaik Mountains and the Bendeleben Range, lying roughly parallel to, and at a distance of 25 miles north of, the important gold fields of the South Coast, rise to heights ranging from 2,000 to 4,000 feet, and afford, as has been proved, a fairly continuous water supply during the open season. The York Mountains, in the western portion of the peninsula, will doubtless furnish a similar water supply should occasion require. Already the extensive ditch systems of Nome River, Ophir Creek, and Solomon River afford an aggregate of 5,000 miner's inches of water in the drier portions of the open season. (See fig. 4.) The expenditure for this work already made will probably not fall far short of a million dollars. Projects for the construction of water conduits are in contemplation which compare favorably with, and even exceed in magnitude, those already completed.

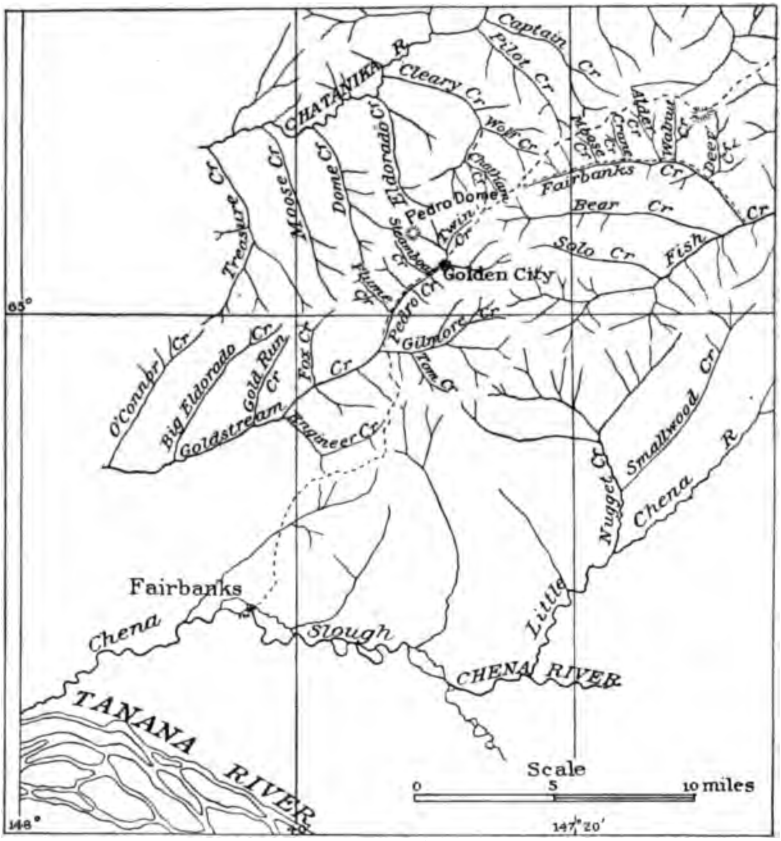


FIG. 3.—Map of part of Fairbanks district.

That the expense attendant on ditch construction in Seward Peninsula is justifiable in certain cases is not to be denied. That the most profitable method of converting the water into power has always been applied is, however, decidedly open to question. The subject will be further discussed under the head of "Hydraulic mining."

The average annual rainfall in California at various points on the west slope of the Sierras is approximately as follows:

| | Inches. |
|---|---|
| Sacramento | 19 |
| Auburn | 20 |
| Colfax | 54 |
| Cisco | 60 |
| Summit | 48 |

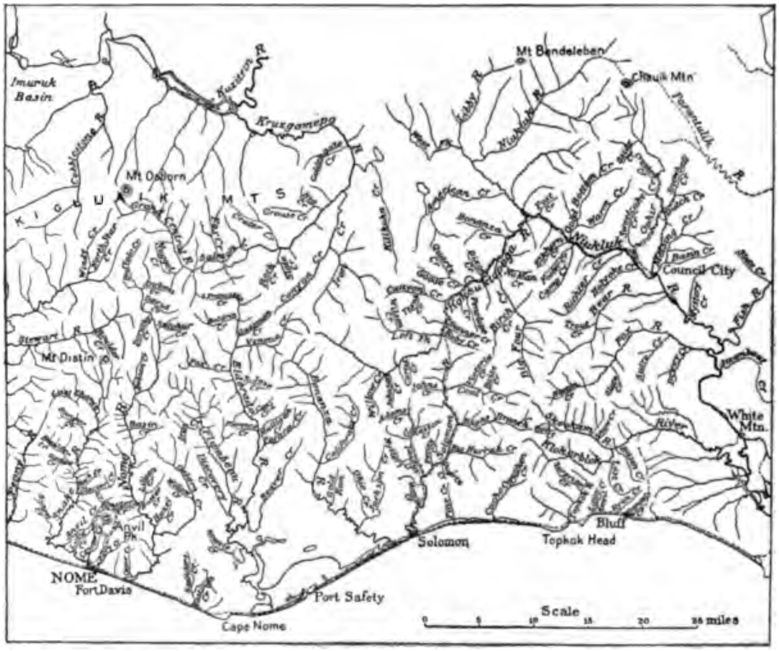


FIG. 4.—Map of part of Seward Peninsula.

Toward the north the rainfall increases in amount, and toward the south it decreases.

Table 2, giving the rainfall in Alaska, shows that there is a difference in rainfall between the coast and the interior gold districts that is too striking to need comment. Comparison of the interior rainfall may be made with that at St. Michael's station, which is the nearest point available representing Seward Peninsula conditions, yet the annual amount of rainfall at St. Michael is still very small compared with that in California.

The accompanying tables, compiled by Cleveland Abbe, jr., on the dates of opening and closing of interior water navigation and of temperature, are here inserted because they have a direct bearing on mining operations. In general, though the fact is contrary to popular opinion, it can be stated that climatic conditions, so far as temperature is concerned, form the least of the hardships with which the miner has to contend. It is safe to say that no more generally healthy climate can be found in any portion of the known world. Data in regard to the length of season available for use of running water will be found in the tabulated list of water conduits. (See table 8, p. 104.)

### TABLE 2.—*Rainfall in Alaska.*

[Compiled by Cleveland Abbe, jr.]

RAINFALL IN INCHES.

| Station. | Jan. | Feb. | Mar. | Apr. | May. | June. | July. | Aug. | Sept. | Oct. | Nov. | Dec. | Year. | Length of record. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | | | | | *Yr.* | *mo.* |
| Juneau | 10.6 | 4.9 | 6.6 | 5.3 | 7.4 | 5.0 | 5.6 | 7.5 | 12.2 | 10.1 | 10.5 | 8.2 | 93.1 | 13 | 11 |
| Skagway | .9 | .6 | .6 | 2.4 | .8 | .6 | 1.7 | 1.5 | 3.5 | 3.2 | 2.2 | 3.8 | 21.8 | 0 | 30 |
| Fort Liscum | 9.7 | 1.0 | 5.5 | 4.5 | 2.3 | .7 | 4.2 | 12.4 | 14.2 | 14.3 | 6.6 | 6.0 | 81.3 | 2 | 0 |
| Kenai | .8 | .7 | 1.0 | .8 | 1.0 | .7 | 1.7 | 3.3 | 2.5 | 2.1 | 1.1 | 9 | 16.6 | 5 | 26 |
| St. Michael | .9 | .2 | .5 | .4 | 1.3 | 1.5 | 2.5 | 3.3 | 4.0 | 1.7 | 1.2 | .8 | 18.1 | 7 | 6 |
| Camp Davidson | .6 | .9 | .2 | .6 | .6 | 2.2 | 1.7 | 3.0 | 2.4 | .2 | .7 | 3 | [13.4] | 0 | 13 |
| Eagle | .4 | .6 | .4 | .8 | .8 | 1.4 | 2.2 | 2.0 | 1.1 | .8 | .6 | | 11.4 | 0 | 52 |
| Circle | ...... | ...... | ...... | ...... | ...... | .5 | 2.0 | 2.7 | ...... | ...... | ...... | | ...... | 0 | 5 |
| Camp Colonna | .7 | 2.1 | .8 | .1 | .6 | .3 | ...... | ...... | ...... | ...... | .4 | 1.2 | > 6.1 | 0 | 8 |
| Fort Adams | 1.7 | .9 | .7 | .4 | 1.5 | ...... | ...... | 2.2 | 1.5 | 2.5 | 2.3 | .8 | >15.5 | 0 | 26 |
| Fort Gibbon | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | 1.6 | .1 | 1.4 | ...... | 0 | 3 |
| Nulato | .7 | .9 | 1.5 | >.2 | .4 | ...... | ...... | ...... | ...... | 1.4 | 1.2 | 1.4 | > 7.5 | 0 | 10 |
| Anvik | 1.3 | .4 | 1.4 | .5 | .6 | ...... | ...... | 2.8 | 2.0 | 1.2 | 1.4 | ..... | >11.7 | 0 | 26 |
| Copper Center | ...... | ...... | ...... | ...... | ...... | ...... | ...... | 1.1 | .7 | 2.0 | 1.5 | .2 | > 5.3 | 0 | 5 |
| Dawson, Y. T.[a] | .7 | .4 | .2 | .78 | .58 | 1.71 | ...... | ...... | ...... | 1.25 | .45 | .65 | ...... | 0 | 19 |

[a] Authority: Inspector of Fisheries, Yukon Territory.

NUMBER OF RAINY DAYS.

| Station. | Jan. | Feb. | Mar. | Apr. | May. | June. | July. | Aug. | Sept. | Oct. | Nov. | Dec. | Year. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Juneau | 18.1 | 11.2 | 18.7 | 15.0 | 16.7 | 14.6 | 15.5 | 15.6 | 18.4 | 19.8 | 18.4 | 19.8 | 201.8 |
| Skagway | 7.5 | 2.5 | 3.0 | 10.5 | 4.7 | 5.0 | 5.7 | 8.5 | 13.5 | 12.0 | 8.0 | 11.7 | 92.6 |
| Fort Liscum | 17.5 | 5.0 | 14.5 | 9.0 | 9.0 | 8.0 | 14.5 | 28.5 | 22.5 | 22.0 | 11.5 | 13.0 | 175.0 |
| Kenai | 5.3 | 4.4 | 7.4 | 6.4 | 6.4 | 5.1 | 8.9 | 13.4 | 11.7 | 10.0 | 7.4 | 6.0 | 92.4 |
| St. Michael | 8.1 | 5.5 | 7.4 | 7.8 | 9.1 | 10.4 | 13.6 | 16.7 | 18.5 | 11.4 | 11.4 | 6.9 | 126.8 |
| Camp Davidson | 16.0 | 14.0 | 6.0 | 9.0 | 7.0 | 11.0 | 14.0 | 16.0 | 16.0 | 6.0 | 13.0 | 7.0 | [151.0] |
| Eagle | 6.2 | 5.8 | 4.8 | 8.6 | 6.5 | 9.5 | 13.0 | 13.0 | 9.8 | 7.8 | 7.0 | 5.7 | 97.7 |
| Circle | ...... | ...... | ...... | ...... | ...... | 4.0 | 8.5 | 9.0 | ...... | ...... | ...... | ...... | ........ |
| Camp Colonna | 14.0 | 16.0 | 12.0 | 4.0 | 4.0 | 5.0 | ...... | ...... | ...... | ...... | 8.0 | 14.0 | > 75.0 |
| Fort Adams | 11.0 | 8.6 | 13.0 | 3.5 | 12.0 | ...... | ...... | 16.0 | 14.0 | 16.0 | 12.5 | 10.0 | >116.0 |
| Fort Gibbon | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | 17.0 | 2.0 | 14.0 | ........ |
| Nulato | 5.0 | 7.0 | 12.5 | >2.0 | 4.0 | ...... | ...... | ...... | ...... | 8.0 | 6.0 | 8.0 | > 52.5 |
| Anvik | 15.6 | 6.0 | 15.3 | 7.6 | 7.0 | ...... | ...... | 13.0 | 11.0 | 12.6 | 13.3 | ...... | >101.4 |
| Copper Center | ...... | ...... | ...... | ...... | ...... | ...... | ...... | 12.0 | 7.0 | 8.0 | 4.0 | 2.0 | ........ |

Numbers in [ ] are totals made up from months in different year

*Numbers preceded* by > are known to be too small.

TABLE 3.—*Average dates of last spring and first fall frosts.*

[Computed by Cleveland Abbe, jr. Dates marked (?) are from a single observation.]

| Place. | Last killing frost. | Last spring frost. | First fall frost. | First killing frost. |
|---|---|---|---|---|
| Juneau | (?) | Apr. 15 (?) | Sept. 20 | Sept. 20 |
| Skagway | Apr. 10 (?) | June 1 (?) | Aug. 30 | Sept. 5 (?) |
| Fort Liscum | (?) | (?) | (?) | Sept. 29 |
| Kenai | (?) | June 15 | Aug. 15 | Aug. 30 |
| St. Michael | June 1 (?) | (?) | Sept. 15 | Sept. 30 (?) |
| Point Hope | (?) | (?) | Sept. 13 (?) | Oct. 15 |
| Camp Colonna | (?) | Apr. 30 (?) | (?) | Oct. 15 (?) |
| Camp Davidson | June 15 (?) | (?) | (?) | Aug. 15 (?) |
| Eagle | May 10 (?) | (?) | Sept. 1 (?) | Sept. 2 (?) |
| Circle | (?) | (?) | Aug. 20 (?) | Nov. |
| Fort Yukon | (?) | (?) | Sept. 30 (?) | Oct. 1 (?) |

TABLE 4.—*Average dates of opening and closing of Yukon and Kenai rivers.*

[Dates marked (?) are from a single observation.]

| Place. | Ice breaks. | River clear. | Ice running. | River closed. |
|---|---|---|---|---|
| Yukon River: | | | | |
| Fort Reliance | May 10 | (?) | Oct. 20 | Nov. 5 |
| Eagle | May 11 | May 15 | Oct. 5 | Nov. 10 |
| Circle | May 13 | (?) | (?) | (?) |
| Fort Yukon | May 15 | May 20 | Oct. 3 | Oct. 26 |
| Fort Gibbon | (?) | (?) | Oct. 25 (?) | Nov. 6 (?) |
| Tanana | May 15 | May 20 | Oct. 15 | Nov. 6 (?) |
| Nulato | May 20 | (?) | (?) | Oct. 20 |
| Anvik | May 15 | May 22 | Oct. 25 | Oct. 31 |
| Kenai River: | | | | |
| Kenai | Mar. 18 (?) | Mar. 29 (?) | Dec. 5 (?) | Dec. 15 (?) |

TABLE 5.—*Table of temperatures, Fahrenheit.*

| Station. | Jan. | Feb. | Mar. | Apr. | May. | June. | July. | Aug. | Sept. | Oct. | Nov. | Dec. | Length record. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | | | | *Yrs.* | *Mos.* |
| Juneau | 34.2 | 33.0 | 37.2 | 41.9 | 46.9 | 51.6 | 54.4 | 56.6 | 52.3 | 45.7 | 39.8 | 36.0 | 5 | [a] 18 |
| Valdez | 23.8 | 15.5 | 30.8 | 31.6 | 39.4 | 49.6 | 50.5 | 46.6 | ...... | ...... | 22.4 | 21.6 | 0 | 10 |
| St. Michael | 7.4 | −2.3 | 8.9 | 19.9 | 33.1 | 46.3 | 53.6 | 51.9 | 43.9 | 30.5 | 15.6 | 4.8 | 11 | [a] 12 |
| Eagle | −24.8 | −6.0 | 13.0 | 29.3 | 42.2 | 52.6 | 56.9 | 49.1 | 40.4 | 20.1 | −10.0 | −7.4 | 1 | 0 |

[a] Nonconsecutive months.

## STREAM VOLUMES.

### THE MINER'S INCH.

Before discussing the matter of stream volumes it will be necessary to consider the unit of measurement, the "miner's inch," which has received so many varying definitions in different mining districts.

On this subject[a] Mr. Hamilton Smith makes the following statements:

> Soon after the discovery of gold in California in 1848 associations or incorporated companies were formed for the purpose of building ditches and storage reservoirs for the supply of water to the placer mines. The amount of capital invested in these hydraulic works aggregated many millions of dollars, and a single company often sold water to hundreds of mining claims. The cost of water was by far the most important item in the miner's bill of costs, and hence it became necessary to have a standard measure, not only accurate but also so simple that the amount of discharge could be readily computed by the common miner.
>
> This was accomplished by the discharge of the stream of water sold to each customer through a rectangular, square-edged, vertical orifice, with free discharge into the air and having a constant head. In different parts of the State the standard opening varied, the width varying from 2 to 4 inches and the head above the top of the opening from 4 to 7 inches. Each square inch of the opening was called "a miner's inch;" hence in a locality where the standard opening was 2 inches wide, if the miner wished a flow of 50 miner's inches, the orifice was 25 inches long, and if only 10 inches was needed the length was reduced to 5 inches.
>
> This method is analogous to the pouce d'eau used in southern France, and was probably first introduced or suggested in California by some French or Mexican miner; the simplicity of this mode of measurement, combined with a sufficient degree of accuracy, soon brought it into general use on the Pacific coast wherever water was sold for mining, irrigation, etc.
>
> The standard which had been in use since 1852 or 1853 in the mining districts supplied by the Eureka-Lake, Bloomfield, and Milton water companies in Nevada County, Cal., was an opening 50 inches long, 2 inches wide, with constant head above opening of 6 inches; the flow from this was called 100 miner's inches. * * * Generally the miners bought water for ten hours per diem at an agreed price per inch; for example, a miner using 350 miner's inches, for ten hours each day, at the rate of 15 cents per inch, paid the water company $52.50 per diem, and received the amount of water which would flow through an orifice having an aggregate length of 175 inches, a width of 2 inches, with a head of 6 inches above the top of the opening, during a period of ten hours.
>
> When water was used for the whole twenty-four hours of the day the flow was termed "a miner's 24-hour inch," and, of course, meant 2.4 times the amount of discharge of "a miner's 10-hour inch."

In California, as larger amounts of water came into use, wider openings were adopted, one being 12 inches high, 12¾ inches long, with a constant head of 6 inches above the top of the opening.

Experiments made in California by A. J. Bowie, esq.,[b] to determine the value of the miner's inch, defined as the one two-hundredth part of the quantity of water which would flow through the last-named aperture in a 1½-inch board under head of 6 inches above the top of the discharge, showed that 1 miner's inch discharged in—

| | Cubic feet. |
|---|---|
| 1 second | 0.02499 |
| 1 minute | 1.4994 |
| 1 hour | 89.9640 |
| 24 hours | 2,159.1460 |

---

[a] Hydraulics, 1886, p. 277.

[b] Bowie, A. J., jr., A Practical Treatise on Hydraulic Mining, 1885, p. 126

Experiments on a single aperture of this form, used in determining the North Bloomfield standard made by Hamilton Smith, jr.,[a] gave a discharge (a mean of two different openings) of 302.7 cubic feet per minute, or 2,179.4 cubic feet per miner's inch in twenty-four hours.

The miner's inch used in this report is one two-hundredth part of the amount of water which will flow through an opening 12 inches high by 12¾ inches wide in a 1½-inch plank, under a constant head of 6 inches above the top of the discharge. This may be taken for all practical purposes as equivalent to 1.5 cubic feet of water per minute, or, in other words, 1 cubic foot per second equals 40 miner's inches.

A simple means of ascertaining the approximate number of miner's inches in an open conduit is to select a straight portion of the ditch or flume where the water runs quietly and where no accelerated velocity has been imparted to it. One hundred and ten feet measured along the bank should be called 100 feet. Floats made by weighting empty cartridge shells with shot or small stones and fitting into them cylindrical wooden plugs 4 or 6 inches long are then placed in the canal as quietly as possible. Note the average time which it takes several of them to traverse the distance, divide the distance in feet by the average time in minutes, and the result will be the velocity in feet per minute; this multiplied by the area in square feet will give the number of cubic feet of water flowing per minute. To get the number of miner's inches, multiply the cubic feet per minute by 2 and divide by 3.

California,[b] Colorado, and Montana have, by State law, made the definitions shown below:

*Value of second-foot (=cubic foot per second) in miner's inches.*

| | Miner's inches. |
|---|---|
| California | 40.0 |
| Colorado | 38.4 |
| Montana | 40.0 |

The measure is of great value and should be defined by Federal law, since at present it is used in an indefinite sense. The placer miner generally assumes, when he is buying water, that he is getting 1.5 cubic feet per minute for each inch, or the inch measured under 6-inch head above the top of the discharge, whereas, in reality, the inch sold him is equal only to 1.2 cubic feet per minute, being measured under a head of 4 inches above the center of an orifice 2 inches wide.[c]

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[a] Hydraulics, p. 282.

[b] Stover, A. P., California Journal of Technology, quoted in Eng. and Min. Jour., May 26, 1904.

[c] The legal definition of the miner's inch in use in the Yukon Territory, as given to the writer by Mr. A. J. Beaudette, the Government mining engineer, is one-twelfth the amount of water that will go through an orifice 2 inches high by 6 inches wide under a constant head of 6¼ inches above the center of the orifice.

## TABLE OF STREAM VOLUMES.

Table 6, below, showing stream volumes in miner's inches, represents an attempt to combine at least approximate data relating to the amount of water which flows in northern auriferous streams during the open season. The few gagings that were made during the course of the trip were by Price current meter. It was found that variations from day to day in all parts of the country were exceedingly frequent, the volume on one day being sometimes from one-third to one-half that of the succeeding day after rain. Where a range between two amounts is given, the smaller figure may be taken as representing the most frequent condition.

TABLE 6.—*Approximate stream volumes, in miner's inches.*

| District and stream. | Gagings. | Volumes in miner's inches. | Time. | Authority. | Remarks. |
|---|---|---|---|---|---|
| Juneau district: | | | | | |
| Gold Creek | | 500-20,000 | Open season. | This report | Low to flood. |
| Atlin district: | | | | | |
| Pine Creek | | 4,000-8,000 | do | do | |
| Spruce Creek | | 1,000 | do | do | |
| McKee Creek | | 500-2,000 | do | do | |
| Boulder Creek | | 1,000 | do | do | |
| Birch Creek | | 500-15,000 | do | do | |
| Klondike district: | | | | | |
| Klondike River | | 55,000-300,000 | do | Canadian government engineer. | |
| Bonanza Creek | | 68-17,641 | do | do | |
| Eldorado Creek | | 820 | do | do | Above mean. |
| Hunker Creek | | 156 | do | do | Low. |
| Bear Creek | | 65 | do | do | |
| Dominion Creek | | 400 | do | do | Mean. |
| Sulphur Creek | | 300 | do | This report | Do. |
| Gold Run | | 120 | do | Canadian government engineer. | Below mean. |
| Eagle district: | | | | | |
| American Creek | 868 | | July | This report | Below forks. |
| Discovery Fork | 220 | | do | do | |
| Circle district: | | | | | |
| Birch Creek | | 70,000 | | do | |
| Deadwood Creek | 582 | | July | do | |
| Mastodon Creek | 322 | | do | do | |
| Mammoth Creek | | 600 | | do | |
| Junction of Eagle Creek and Mastodon Fork. | 255 | | July | | |
| Rampart district: | | | | | |
| Minook Creek | | 1,000-10,000 | Open season. | Members of Geological Survey. | |
| Little Minook Creek | | 50-1,000 | do | do | |
| Little Minook Jr. Creek | | 0-200 | do | do | |
| Hunter Creek | | 600-4,000 | do | do | |
| Hoosier Creek | | 500-3,000 | do | do | |
| Ruby Creek | | 200-2,000 | do | do | |
| Slate Creek | | 200-2,000 | do | do | |
| Eureka Creek | | 100-1,000 | do | do | |
| *Pioneer Creek* | | 200-1,000 | do | do | |

TABLE 6.—*Approximate stream volumes, in miner's inches*—Continued.

| District and stream. | Gaggings. | Volumes in miner's inches. | Time. | Authority. | Remarks. |
|---|---|---|---|---|---|
| Rampart district—Cont'd. | | | | | |
| Glen Gulch | | 0–500 | Open season. | Members of Geological Survey. | |
| Gold Run | | 10–200 | do | do | |
| Rhode Island Creek | | 50–500 | do | do | |
| Seattle Creek | | 50–500 | do | do | |
| Omega Creek | | 25–500 | do | do | |
| Thanksgiving Creek | | 10–500 | do | do | |
| Fairbanks district: | | | | | |
| Fairbanks Creek | 117 | | August | This report | Low water. |
| Cleary Creek | 400 | | do | do | After rain. |
| Wolf Creek | | 25 | do | do | |
| Chatham Creek | 10 | | do | do | |
| Pedro Creek | 110 | | do | do | Below Twin Creek. |
| Gold Stream | 450 | | do | do | Junction of Gilmore and Pedro. |
| Nome district: | | | | | |
| Anvil Creek | | 100–400 | Open season. | Members of Geological Survey. | |
| Glacier Creek | | 300–1,000 | do | do | |
| Dexter Creek | | 2–200 | do | do | |
| Nome River | | 12,000–20,000 | do | do | |
| Snake River | | 8,000–18,000 | do | do | |
| Bourbon Creek | | 10–100 | do | do | |
| Dry Creek | | 25–500 | do | do | |
| Osborn Creek | | 500–2,000 | do | do | |
| Buster Creek | | 100–1,000 | do | do | |
| Basin Creek | | 250–2,000 | do | do | |
| Rock Creek | | 10–50 | do | do | |
| Council district: | | | | | |
| Ophir Creek (mouth) | | 3,000–20,000 | do | do | |
| Melsing Creek | | 200–1,000 | do | do | |
| Crooked Creek | | 50–200 | do | do | |
| Warm Creek | | 100–500 | do | do | |
| Gold Bottom Creek | | 800–4,000 | do | do | |
| Mystery Creek | | 100–500 | do | do | |
| Solomon district: | | | | | |
| Solomon River | | 6,000–50,000 | do | do | |
| Big Hurrah Creek | | 500–2,000 | do | do | |
| Shovel Creek | | 1,000–5,000 | do | do | |

## GRADES OF STREAMS.

No subjects are of greater importance for the consideration of the placer miner than the grade available for moving his gravel and the vertical space which he can obtain for a tailings dump. Neglect to carefully consider the natural grade of the ground on which the proposed operation is to be undertaken will generally result in financial failure.

Alaska is not characterized by steep grades in the parts which are rich in alluvial gold. In fact, with the exception of the relatively small placer districts of the South Coast province, the long-continued conditions of erosion have produced surprisingly low gradients in the streams. Table 7, below, gives the grades of streams in the gold-producing districts of the north. For the sake of showing the defects of Alaska in the important requirements of grade, the lowest figure in the table gives what is frequently referred to by gravel miners as "sluice-box grade." If the sluice boxes are carried on a grade lower than 6 inches to the 12-foot box, the gravel can not be economically moved through it except by employing an excessive quantity of water. By comparing the sluice-box grade with those of the various Alaska creeks, it becomes evident that in nearly every case the material, either before entering or after leaving the tail sluice, must be artificially elevated in order that the sluice may have sufficient grade. The various devices in use for accomplishing this elevation will be discussed under proper headings.

TABLE 7.—*Stream grades along important sections of their courses.*

| | Average fall in feet per mile. |
|---|---|
| JUNEAU DISTRICT: | |
| Gold Creek | 292 |
| NIZINA DISTRICT: | |
| Dan Creek | 200 |
| ATLIN DISTRICT: | |
| Pine Creek | 50 |
| McKee Creek | 430 |
| KLONDIKE DISTRICT: | |
| Eldorado Creek | 50 |
| Bonanza Creek | 45 |
| Gold Run | 50 |
| FORTYMILE DISTRICT: | |
| Wade Creek | 75 |
| Chicken Creek | 80 |
| Walker Fork | 100 |
| EAGLE DISTRICT: | |
| American Creek | 90 |
| CIRCLE DISTRICT: | |
| Deadwood Creek | 70 |
| Mastodon Creek | 75 |
| Mammoth Creek | 60 |
| Eagle Creek | 100 |
| FAIRBANKS DISTRICT: | |
| Fairbanks Creek | 75 |
| Chatham Creek | 250 |
| Cleary Creek | 60 |
| Pedro Creek | 65 |
| RAMPART DISTRICT: | |
| Little Minook Creek | 90 |
| Hunter Creek | 75 |
| NOME DISTRICT: | |
| Anvil Creek | 100 |
| Glacier Creek | 50 |
| Dexter Creek | 120 |

| | Average fall in feet per mile. |
|---|---|
| COUNCIL DISTRICT: | |
| Ophir Creek | 25 |
| Melsing Creek | 50 |
| SOLOMON DISTRICT: | |
| Solomon River | 20 |
| Shovel Creek | 90 |
| AVERAGE SLUICE-BOX GRADE | 220 (or 6 inches to 12 feet). |

## OPEN-CUT MINING.

### INTRODUCTION.

Under this heading will be discussed the various methods of mining by which gravel is taken out of open cuts—either simple pick-and-shovel methods or methods involving the use of mechanical contrivances. Hydraulic methods will be considered in a later section. The simplest of the open-cut methods dominated throughout the province up to within a few years, to the practical exclusion of all others except winter drifting, which will be considered below. To-day probably 60 per cent of the placer-mining operations in Alaska are confined to open-cut methods, and this investigation has brought the proof that, when combined with proper mechanical devices, these form often the most economical mode of exploitation.

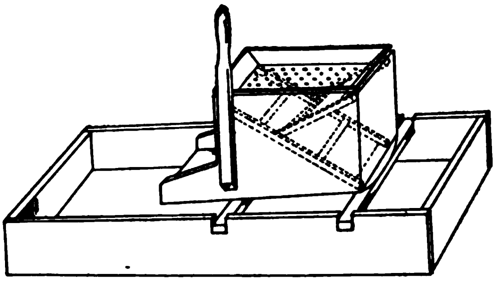


FIG. 5.—Klondike rocker.

Open-cut methods can be conveniently grouped under various headings, but the underlying principle of excavating and transporting the material to the washing apparatus, either by hand labor or by some mechanical means, remains the same. It therein differs from hydraulic mining, where the gravel is moved by water under pressure. There are, of course, various intermediate processes, which will not require separate consideration.

### ROCKER AND LONG TOM.

The simplest method of work which the miner adopts after he has passed the stage of panning is that of shoveling from the bank and washing the gravel in a rocker. The use of the rocker is too well known to need description. The form employed in the Klondike, where its use has been nearly discontinued in mining, is shown in fig. 5. Two men are necessary to use the rocker properly, while only 3 to 5 cubic yards of gravel can be washed in ten hours.

At no place was the long tom seen in use in the north, although it was formerly employed to some extent in washing the Nome beach placers.

## SHOVELING INTO SLUICE BOXES.

### CONDITIONS FOR SHOVELING.

Conditions throughout many of the northern placer districts favor this well-known, simple, and cheaply installed method of placer mining, for in many localities the pay streaks are thin, ranging from 2 to 4 feet in thickness, rarely exceeding 5 feet. It is often applicable where conditions of transportation prohibit the installation of more elaborate plants.

When the total depth to the bottom of the pay streak does not exceed 12 feet, the overlying barren material can generally be "ground sluiced" off, even where the grade does not exceed 1 per cent, at an expense varying from 7 to 20 cents per cubic yard. The frozen muck which composes this overburden is from 50 to 75 per cent water, and the solid residue of silt or fine clay is easily carried away at any time of slight rise in the stream. The method of ground-sluicing the muck is described under the heading "Hydraulic mining," on page 141. Pl. IV, *A*, shows the ordinary method of setting up a string of sluice boxes on a creek placer. This method is used in an elaborate form on Anvil Creek, where 5 strings of boxes and 120 shovelers give a twenty-hour capacity of 1,080 cubic yards.

The handling of the water is important. Even when the gravel has been solidly frozen previous to stripping and sun thawing, there are generally from 15 to 25 miners' inches of seepage water, which work into the pit floor. This must be disposed of either by draining or by pumping. If sufficient water (50 miners' inches) is available, a "China pump" can be rigged up, as shown in Pl. IV, *B*. This device is not economical in its use of water, for the water is generally of more value for other purposes in Alaska mining.

### DAMS.

As in all other forms of creek mining, great care must be taken to keep the creek water out of the working pits by means of dams. Dams constructed of sod walls lined with sacks have been found cheaper in the Klondike than those built of sod and brush. Pile driving is not advisable in the north, but in case timber is abundant cribbing is desirable, as illustrated in Pl. V, *B*, showing a dam built across Discovery Fork of American Creek to impound water for "booming." The ends of the logs were set in frozen ground on both sides and the muck was allowed to refreeze around them. The dam, 40 feet long, consists of 12-inch timbers, from 9 to 18 feet long, laid up in two rows, 5 feet apart, earth and rock filled, and braced with

A. SET-UP OF BOXES FOR SHOVELING-IN, AMERICAN CREEK

B. CHINA PUMP, KLONDIKE.

cross timbers as indicated. The dam, self-dumping gate, and accessory flumes were built by two men, and are said to have cost, in labor and time, only $300.

The self-dumping gate shown in Pl. V, *B*, was not in use when the visit to Discovery Fork was made. It had been used during the month of June, 1904, in "booming" the overburden from the area of pay ground which it was desired to uncover. The action of the gate is apparent from the illustration. Swinging on its horizontal axle, set two-thirds the distance from the top to the bottom of the gate, it releases the water when the reservoir becomes full and regains its closed position when the reservoir is nearly empty. During a period of three weeks a block of overburden consisting of muck and barren gravel, 5 feet thick by 25 feet wide by 900 feet in length, had been removed by the booming process before the shoveling operations commenced, the total cost, including the construction of the dam and gate, not exceeding 7 cents per cubic yard.

The process of booming by means of automatic water gates was formerly generally in use in California, and is employed even to-day in southern Oregon. Pliny mentions the process as having been in extensive use in Spain before the Christian era. As a rule it is inapplicable to mining or stripping in Alaska, because the stream grades are gentle and because the débris carried down by the torrential stream will damage property situated farther down the creek.

On Hunker Creek, in the Klondike, a dam of moss, brush, and gravel, 90 feet long and 18 feet high, built for the purpose of keeping the water from an open cut, cost $500. Dams in Seward Peninsula are successfully built of sod, the material which overlies the frozen muck to the depth of 2 feet. Sacks of sand are also successfully used.

### DRAINS.

Below the proposed pit a backwater dam, generally 4 feet in height, is built across the creek, and the end of the sluice is extended beyond this dam a few feet. Pl. IV, *A* (p. 56), shows the system of damming to keep the water out of a small shoveling pit, and Pl. V, *A* (p. 58), shows backwater dam and the standpipe erected to receive water for the bed-rock drain. Assuming that the ground is to be worked upstream, the method of laying off the drain, commonly called the "bed-rock drain," is as follows: In 12-foot ground take a level and sight downstream from the prospect pit to a point where 12 feet vertical distance above the water in the stream is obtained. The sight should be taken at a point 2 feet above the surface of the ground, to allow for the height of the backwater dam. The point where the drain is to be started is the distance downstream obtained by the sight plus 12 feet, allowed as a safety factor. On a 2 per cent grade this would necessitate a drain 612 feet long. The drain is dug and left open until after the first cut is made. Its dimensions are generally

2 by 2 feet and it is lagged inside with poles horizontally, these being held by posts with 4-foot centers, into which shoulders are cut for caps. After the first cut the drain must be covered by logs and moss laid on top of the caps. For small operations a box drain made of sawed 1-inch lumber, the inside dimensions being 12 by 10 inches, is used. There should be a standpipe bored with auger holes in the drain at the lower end of every cut. The drain should have a very slight grade, as low as 1½ inches to 100 feet being used. At first the drain will handle only seepage water, but if one cut of 100 feet in length after another is successively worked upstream, sluice water may have to be run through it to prevent backing up. Some operators recommend keeping the main drain large and giving it the bed-rock grade, while a smaller drain, 8 by 8 inches, is kept parallel to it on a less grade. This will handle the seepage water and will gain depth upstream, thus allowing for inequalities in the bed rock, which are very frequent.

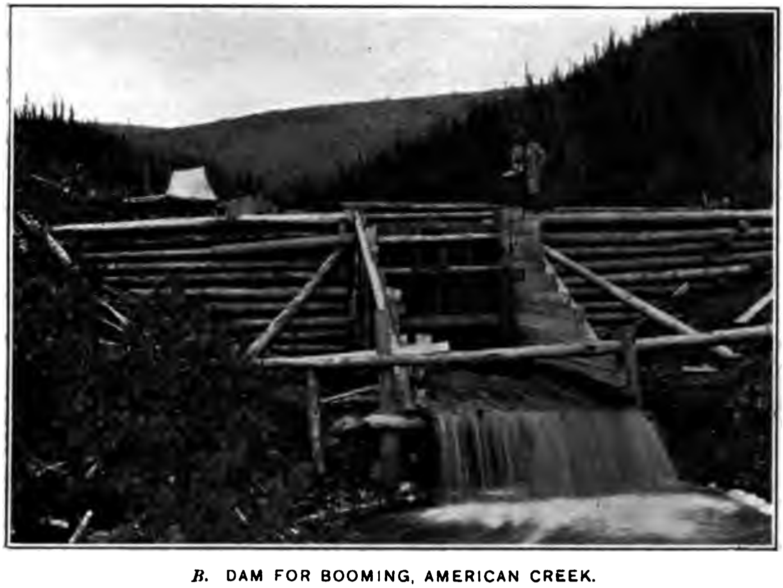
Drains are expensive to make but are generally preferable to pumps. On Mastodon Creek, in the Birch Creek district, a bed-rock drain, 500 feet long, dug 4 feet wide, lagged inside with 4-foot cord wood, making it 2 feet in the clear, and covered with poles and moss, was dug through cemented and frozen gravel and bed rock, and cost $7,000, its construction consuming all of one season. In the Fairbanks district an open drain, not lagged, 3 feet deep and 3 feet 6 inches wide, took the labor of 2 men six weeks, costing nearly $900.

In excavating for drains the ground should be cut in terraces, so that when it commences to thaw it will not run and clog the canal. Pumping seepage water from the pit is to be condemned in general as strongly as pumping water for sluicing. In Bonanza Creek, Klondike, an operation involving the handling of several thousand yards was said to be more expensive by 40 cents per cubic yard when pumping of the seepage was done than when the water was handled by drain. The pumping of seepage water by any form of pump may be estimated to add at least 25 cents per cubic yard to the expenses of handling the gravel. The use of overshot wheels operating China pumps is cheap where water is plenty. The small one illustrated by Pl. IV, *B*, was using 20 inches of water to lift about one-third this amount a height of 10 feet. Such a pump in the Klondike, with 5-foot wheel, costs $100 to build.

## COST OF SHOVELING.

The cost of a plant, including that of constructing a seepage-water drain, sod dams, and a string of 10 sluice boxes, with a ditch to lead the water to the sluice, varies from $500 to $2,000, according to the conditions. As stated in the cost sheet (table 1, p. 38), the expense of handling gravel by this method is from $1.25 to $2.50 per cubic yard, reaching $5 to $7 in remote districts.

*A*. YUKON PLACER MINER.



*B*. DAM FOR BOOMING, AMERICAN CREEK.

The duty of a man shoveling is variously estimated in different districts of the North. This form of mining having been nearly discontinued in the Klondike region, very little information concerning it was obtainable. The extensive shoveling operations on upper Bonanza Creek were not visited. In one case, where a platform was used, 2 men shoveling, in two stages, a total of 9 feet lift, 2 feet of pay were shoveled into the boxes at the rate of 3½ cubic yards per ten-hour man.

In the Birch Creek district the duty of a man shoveling from an average of 12 operations is 5 cubic yards in ten hours, the depth of pay averaging 4.41 feet, and the lift never exceeding 6 feet.

In American Creek, where the bowlders were large and men had to stop work to get them out of the way, the duty was 2¾ cubic yards. On Discovery Fork of this creek the duty was 4 cubic yards, the depth being 5 feet.

In the Fairbanks district, where shoveling-in operations are few, owing to the depth of ground, the average of three seen was 7½ cubic yards duty, the depth shoveled being 3 feet of pay, part gravel and part bed rock. In no case was there a lift of over 5 feet.

In the Nome region 5.76 cubic yards duty is probably very near the average on a bank from 5 to 7 feet in height. In the case of the large operations of the Pioneer Mining Company, on Anvil Creek, the work has been so systematized that on the 3-foot bank handled the duty is 9 cubic yards per ten-hour man.

One operation on Solomon River, where the lift was high, and the bed rock irregular, 3.75 cubic yards was the duty on a 3-foot bank.

In the Council district of Seward Peninsula, including operations on Ophir, Crooked, Warm, Gold Bottom, Camp, and Penelope creeks, the duty of a man averages 6.63 cubic yards on an average height of bank of 3.5 feet. This average is affected by one case where on a limestone bed rock, with double lift, the duty was 3.5 yards; also by another where with eight-hour shifts, working on 3-foot bank with 5-foot lift, the extraordinary duty of 12 cubic yards per man was vouched for.

The method of working shallow placers by shoveling into boxes has much to commend it, especially when the water is drained from the cut, either with or without a covered drain. In Alaskan gravels exceeding $3 in gold tenor to the cubic yard of material handled by shoveling, frequently from one-half to two-thirds of the gold lies in the crevices of the upper 18 inches or 2 feet of the bed rock. Men directly shoveling this material can thoroughly clean the bed rock by the shoveling-in method. On the other hand, if horse scrapers, steam scrapers, or steam shovels are used, the bed rock is frequently not cleaned, and a gang of men must go over the ground a second time to pick up the auriferous material which has been left. This point is of the first importance. No operator should contemplate the installation of mechanical excavators for working gravel without taking full account of it.

In considering methods of mining by water or mechanical power, the cleaning of bed rock by men may be permissible from an economic standpoint in exceptional instances, as, for example, where a thick overburden is cheaply removed by hydraulicking and a very thin rich pay streak remains. When, however, a mechanical method of removing overburden and low-grade gravels costs from 50 cents to $1 a yard, and the whole area has to be gone over again by men to wheel the rich pay to the sluice, the gravel must be extraordinarily rich to pay a profit. The lowest cost of handling gravel by the method of shoveling in was found to be $1 per cubic yard on one of the creeks of Seward Peninsula, while the cost may reach $5 on some of the isolated interior creeks of Alaska.[a] From 10 cents to $1 per square yard of area worked must be generally added to the shoveling cost to cover the cost of stripping muck or overburden. Ground which exceeds 12 feet in depth of combined stripping and pay will rarely pay if handled by the method of shoveling into the sluice.

## HORSE SCRAPING INTO SLUICE BOXES.

Ground which can be worked by men shoveling into sluices can, under certain conditions, be worked satisfactorily by horse scraping, and at an expense of one-third of that necessary to shovel in. The most important governing condition is the degree of looseness in the gravel and in the underlying auriferous bed rock. Two horses, or, better, mules, hauling a scraper, with driver, will cost from $17 to $23 a day, and in the ordinary small gravel of the Alaska placers, with soft schist bed rock, the team will scrape into the boxes from 30 to 40 cubic yards of gravel a day over a distance of 75 feet. On Penelope Creek, in Seward Peninsula, it was said that the team would handle as much as 10 men could shovel, the cost per cubic yard being 30 cents.

One common breaking plow with a team of horses suffices to break up enough ground for four scrapers. The method requires an inclined platform built up to a height of 10 to 15 feet above the bed rock, over the head box of the sluice. A rectangular opening in this platform serves as a chute to the dump box. Generally the horses travel in an elliptical track, passing the end of the tail box and scraping the tailings from it, then entering the pit, scraping up the pay, and afterwards delivering it to the sluice. In Seward Peninsula the cost of this method can be brought as low as 25 cents per cubic yard, and in no part of the interior will it exceed 50 cents, exclusive of top stripping.

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[a] See cost sheet, table No. 1.

Many deposits were seen now worked by shoveling into sluices where at least an experimental trial of this method is to be recommended. One of the strongest points in its favor is the cheapness of installation and the mobility of the apparatus used.

## STEAM SCRAPERS.

In view of the economical showing which the steam scraper has made in handling tailings from the creek-mining operations of the Klondike it is remarkable that so few attempts have been made with it to scrape the alluvium from its original position. The ordinary scrapers used in steam-scraping operations on tailings in the Klondike have a capacity of from one-third to one-half cubic yard, operated by double drum, 2-cylinder hoist; 16-inch drums, 25 to 30 horsepower capacity, handling on an average 250 cubic yards of loose material in twenty-four hours, at an average expense of 49 cents per cubic yard. The manipulations of the scraper, considered as a unit of the plant, take 3 to 4 men on shift—a fireman, a hoistman, and either 1 or 2 men to fill, guide, and dump the scraper. The form and rigging up of the scraper, with the system of sheaves, pulling, and drawback cables, is shown in figs. 6 and 7. In the practice seen the scrapers drag the material from the pit to the dump, a distance measured horizontally from 100 to 300 feet and vertically from 20 to 50 feet. The scrapers are not always provided with teeth like the one shown in the figure, but this is advisable. A rigid bale should never be used, as flat stones catch between it and the body of the scraper. The plants average $3,500 in cost.

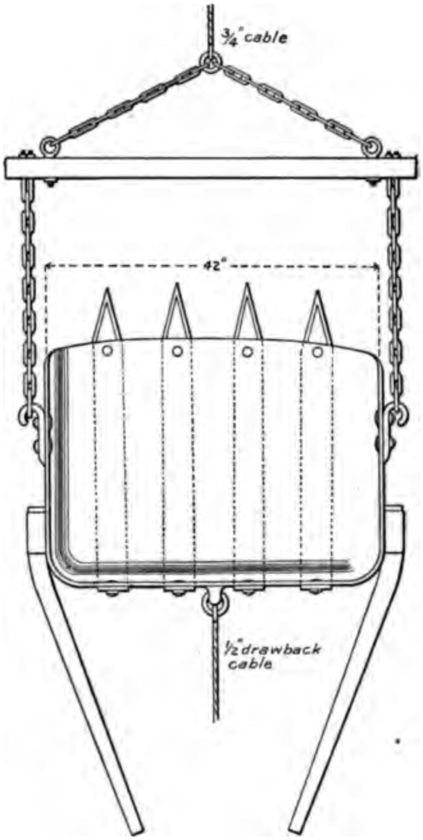


Fig. 6.—Small steam scraper, Klondike.

The rig shown in Pl. VI, *A* (p. 62), is situated on Walker Fork, Fortymile district, Alaska, where a body of gravel 60 feet in width and 5 feet in depth, with little or no stripping, was being mined. According to information obtained by Mr. L. M. Prindle the gravel was only partially free, and little or no stripping was required. As a rule the scraper was found to clean the schist bed rock satisfactorily. The scraper used had a capacity of one-fifth cubic yard, being an ordinary horse scraper rigged for steam with pulling and drawback cables. A 6-horsepower hoist operated the scraper, giving a capacity of 100 cubic yards in twenty-four hours. The 10-horsepower boiler also furnished steam to operate the bucket elevator which lifted the gravel to the sluice from the hopper, to which it was dumped by the scraper. A conservative estimate places the cost of handling gravel with rig, 3 men being employed and one-half cord of wood burned in ten hours, at 40 cents per cubic yard. The operations were meeting with financial success.

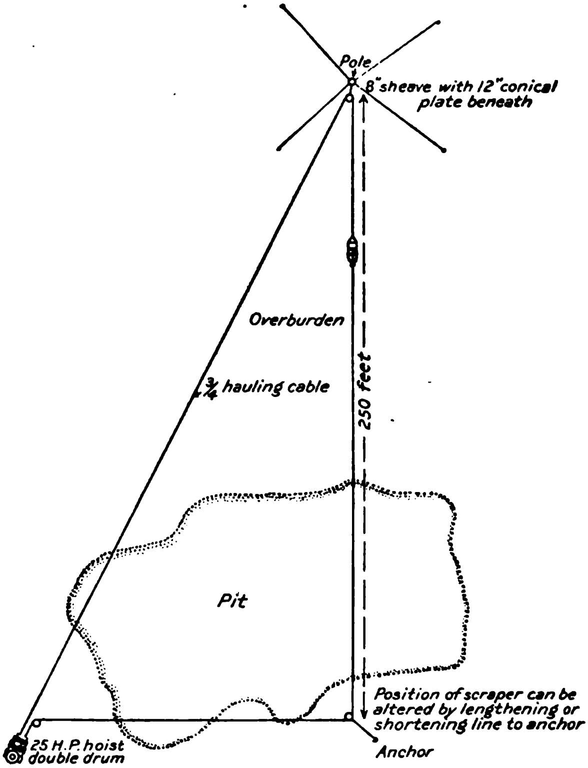
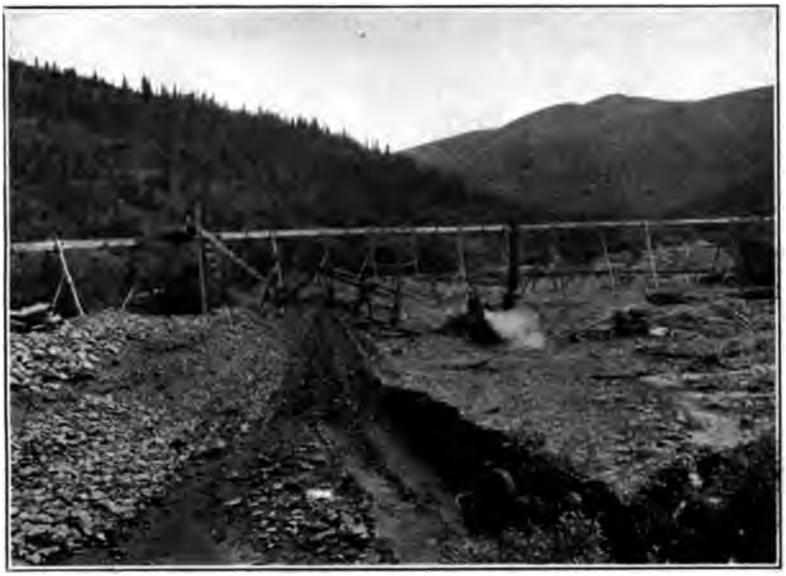
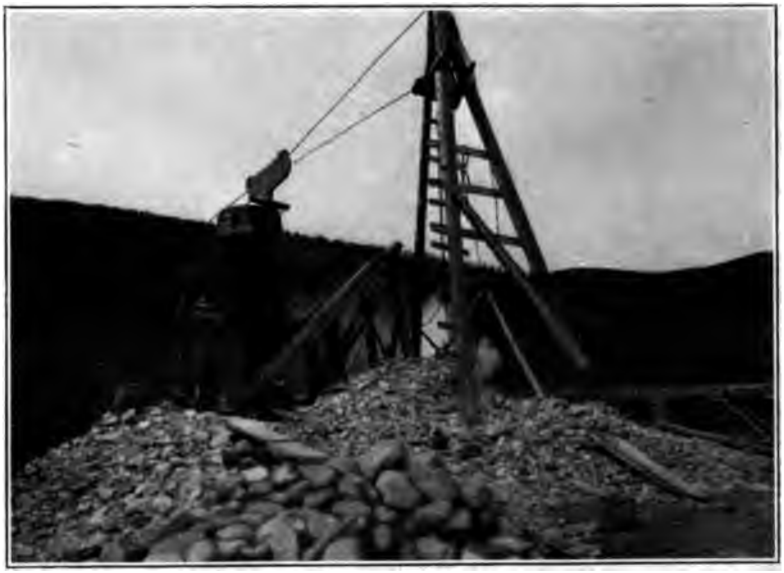

Pole
8" sheave with 12" conical plate beneath
Overburden
¾ hauling cable
250 feet
Pit
Position of scraper can be altered by lengthening or shortening line to anchor
25 H.P. hoist double drum
Anchor

FIG. 7.—Set-up of scraper, Klondike.

The plant contains the elements of a device which, it is not unlikely, may be applied to the working of the wide, shallow creek deposits of Seward Peninsula, or of such creeks of the interior as Mammoth, in the Birch Creek district, or lower Pedro, in the Fairbanks district. If circumstances warranted, the sheave anchor (a rock-filled crib) shown in Pl. VI, A, could be made more easily movable by mounting *it on a truck* running on track, it being made fast by cable and dead-



A. STEAM SCRAPING ON WALKER FORK, ALASKA.



B. DUMPING TO SLUICES, KLONDIKE.

man when occasion might require. The bucket elevator employed is not recommended, as the use of such contrivances for handling gravel, unless they are specially and expensively constructed—as on large dredges—is condemned by experience. In place of this, the scraper should be dragged entirely to the point of final discharge to the sluice on an inclined platform. The operation will necessitate the building of a more elevated sluice or washing plant, surmounted by a hopper, the gravel being fed from this in order that the feed may be as nearly continuous as possible. Assuming that the elevation of the head box of the sluice shown in the illustration is 12 feet above the surface of the ground, the scraper could be dragged to double this elevation at a cost not exceeding 10 per cent above that necessary to haul it up to the 12-foot elevation. One of the types of washing plants described on pages 193 et seq. would be applicable with modifications to a steam scraping plant.

It may be well to consider here the reasons why, in wide, shallow creek deposits, steam scraping appears to be preferable to other mechanical methods. In general, it may be said that this method is applicable to operations on schist bed rock, but is entirely impracticable on limestone or other hard bed rock. This is also true of the steam-shovel and dredging methods. Grant that bed-rock conditions are suitable, and consider a solidly frozen deposit of characteristic small gravel with muck and moss overburden, 150 feet width of pay and 7 feet in total depth from grass roots to bottom of pay. After the moss is plowed up, the 3 feet depth of muck can be ground sluiced off in the ordinary manner at an expense of, say, 15 cents per cubic yard. The ground sluicing should be done at the earliest possible moment in the open season or, preferably, late in the fall. The operator now has to deal with a stripped block of frozen gravel and auriferous bed rock 4 feet in depth, carrying pay, or the material which it is desired to sluice. A vertical bank of this material can not be attacked by any mechanical appliance yet devised, but if left uncovered for six weeks it will thaw to bed rock. The operator, however, wishes to take advantage of the continual thawing of the surface by the sun, and to do this he must attack not a vertical but a horizontal surface. The rate of natural thawing varies from 6 inches to 1 foot a week. The scraper armed with teeth rigged so as to make a series of transverse cuts, say for 300 feet lengthwise of the channel, will attack the ground, scraping off the thawed material better than any other mechanical appliance.

The sluice boxes or washing plant built to a height of 25 feet above the surface of the ground can now be approached by a broad incline platform, or, if desired, the scraper may dump to cars which convey the material by gravity to an isolated and conveniently situated washing plant. The system adopted will depend on the magnitude of the operations. In only very exceptional cases should the expense of handling, after the material is once dumped from the scraper, be allowed to exceed 5 cents to the cubic yard.

The plant above suggested implies the use of a scraper of large capacity and should handle the gravel at a total cost, including depreciation charge and added cost of stripping, not to exceed 40 cents per cubic yard in Alaska, provided that 700 cubic yards of gravel a day are sent to the sluice. In order to accomplish this a scraper, preferably of the bottomless, self-dumping type, of 6 yards theoretical capacity, should be used. It will be found that actually 3½ yards will be delivered each time by such a scraper. The operations will require a 60-horsepower boiler and double-drum hoist, and the services of seven men on a shift. The total running expense of the plant should not exceed $150 a day of 24 hours.

The double-drum hoist operating the scraper will have a position on the side of the sluice opposite that of the cut, and can be mounted on skids so that it can be easily moved by means of a sheave and deadman if necessary. The sheave through which the drawback cable runs may be anchored to a weighted car running on 200 feet of track laid parallel with the cut on the side opposite that occupied by the sluice and hoist. If it is desired to keep the drawback cable out of the way it may pass through two sheaves, one anchored to traveling anchor and one to deadman, the cables thus forming a triangular arrangement, two of whose angles will vary as the car is moved to cover various parts of the ground. The car may be moved as often as desired, and thus one furrow after another may be made by the scraper over a triangular area as the ground thaws. By means of a rearrangement of the sheaves it is found that nearly all of the ground can be covered. A plant erected for the Klamath River Gold Mining Company[a] is provided with two 1¼-yard scrapers, which travel back and forth alternately, both cables acting as pulley and drawback cables. Two sheaves are used on the side of the excavation opposite to the washing plant. "These sheaves are attached to a spreader which keeps them spaced a given distance apart, and to each end of the spreader is attached a tackle which runs back at an angle to deadmen to which they are securely anchored."

The type of bottomless scraper shown in fig. 8 (p. 65) was seen in successful operation stripping loam at a reservoir excavation near Portland, Oreg. It has a theoretical capacity of 6 yards, and actually handles a little over half this amount. Measurements of a spoil bank showed that in seven ten-hour days, stripping to 4 feet in depth, 400 cubic yards per shift had been handled. The scraper was making furrows over 300 feet in length. A 60-horsepower boiler was used,

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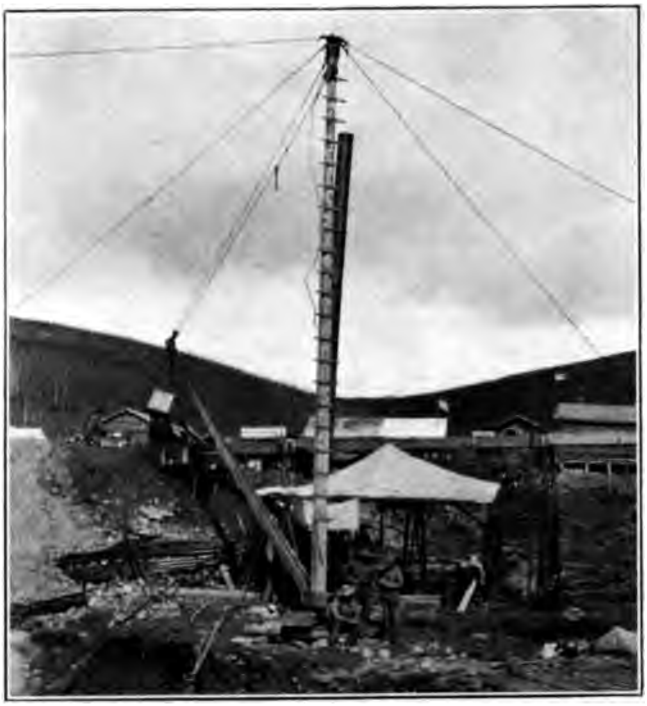
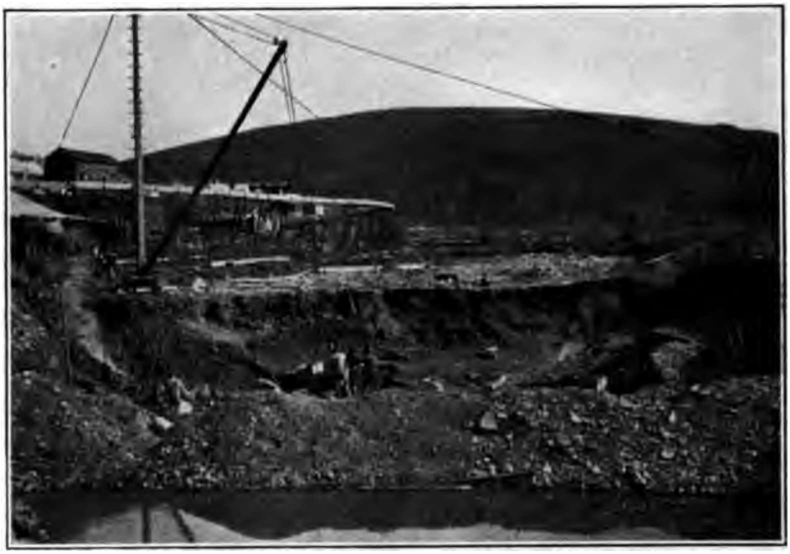
[a] Yeatman, J. A., Automatic excavator for placer mining: Min. and Sci. Press, Dec. 17, 1904.



*A.* DERRICKING AT FAIRBANKS, DUMPING.



*B.* DERRICKING AT FAIRBANKS, LOADING.

but only one cord of wood at $2 per cord was burned. The double-drum hoist was provided with 10 by 12 inch cylinders and was geared 6 to 1. Four men, a winchman, a fireman, and two scraper men were employed, at $2.50 a day. It was said that under these conditions the operations cost 5 cents per cubic yard. It was estimated that in a haul of from 150 to 200 feet the scraper would deliver loam to the spoil heap at the rate of 2 cubic yards per minute.

The above suggestions regarding the use of self-dumping scrapers in northern placer mining will doubtless be looked on with considerable skepticism. No direct application of the method has, so far as known, been made in Alaska. The form of plant outlined is inexpensive as compared with many already installed in Alaska, having a capacity not exceeding 700 cubic yards. The sum of $10,000 should be ample to install an entire scraper and washing plant in Seward Peninsula. Even should some form of steam scraping be found applicable to the wider shallow deposits, the installation of elaborate cableways, traveling towers, and the like, is not advisable, as in most cases their cost would be prohibitory.

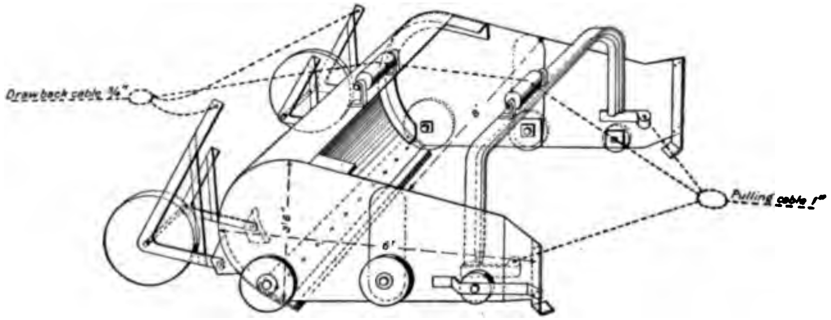


FIG. 8.—Bottomless steam scraper.

### SELF-DUMPING CARRIERS.

The conditions in the Klondike district appear to have necessitated the adoption of this expensive method of placer mining. Outside the Klondike field the method was not seen except at one place on Mastodon Creek, in the Birch Creek district. The method is adopted to work rich gravels where conditions do not permit working by ordinary shallow open-cut methods, and where drifting is impossible or inadvisable.

In considering the economic success of this method, a study of the cost sheet given in this report (table 1, p. 38), is most instructive. In the first place the average depth from eight Klondike operations considered, namely 17.5 feet, is greater than that economically advisable (namely, 15 feet) for open-cut operations in general. The operator adopts this method because there are thawed streaks and channels in his deposit. If such ground is drifted, the chances are that he will increase his expense to a prohibitive amount through excessive timbering and through pumping of seepage water, whereas by the open-cut method the water is handled by drain. The cost sheet shows that whereas $2.14 is the average cost of the above-mentioned 8 operations in the Klondike, 7 drifting operations in similar deposits, with an average depth of 25.3 feet, gave an average cost of $1.95 per cubic yard, the depth of the pay or thickness of gravel actually sent to the sluice being almost exactly the same in both cases, and the capacity in the drifting being only 17.5 cubic yards less in twenty-four hours. Granting, however, that in a given deposit carrying $3 to the cubic yard of pay, the depth being 16 feet, drifting is impossible, and the rich pay, 75 feet in width, must be worked by open cut. Whatever method be adopted the moss must first be plowed up and about 6 feet of muck ground sluiced off at a cost of 17 cents per cubic yard. Next 6 feet of barren gravel or sand must be removed, either by horse scrapers at 60 cents per cubic yard, or, if the plant warrants the operation, by steam scraper at 49 cents per cubic yard. The 4 feet of pay being laid bare, what method shall be adopted to get it into the sluices? On account of the necessarily short life of the operations, a plant whose first cost exceeds $5,000 is out of the question. The greatest expense will then result from the hand shoveling in the pit. The getting of the material into the receptacle in which it is conveyed to the sluice is the principal item of expense in the operation. It is therefore necessary that the high-priced shovelers get as much gravel into the wheelbarrows or buckets as possible. The bucket, 37 inches square on top, 35 inches square on bottom, and 25 inches deep, holding two-thirds of a cubic yard, is dropped into a crib built in the bottom of the pit, to which the shovelers wheel their dirt in wheelbarrows. From 4 to 6 wheelbarrows are necessary to fill the bucket. There is no mobility to the bucket; it must always rise and fall to the same spot. Men instead of occupying all their time in shoveling are employed nearly half of it in wheeling and dumping. Five operations are necessary to get the gravel from the bank to the sluice, namely, (1) shoveling into wheelbarrows, (2) wheeling to bucket, (3) dumping to bucket, (4) raising bucket to carrier.[a] (5) conveying and dumping to sluice. (See Pl. VI, *B*.)

This may now be compared with the derricking system. On Pedro Creek, Fairbanks district, an open cut 15 feet of depth with 9 feet of pay gravel is worked by derricking. The plant has a capacity of 233 cubic yards a day, which is handled at a cost of $1.75 per cubic yard. The plant costs no more than the average price of a self-dumping

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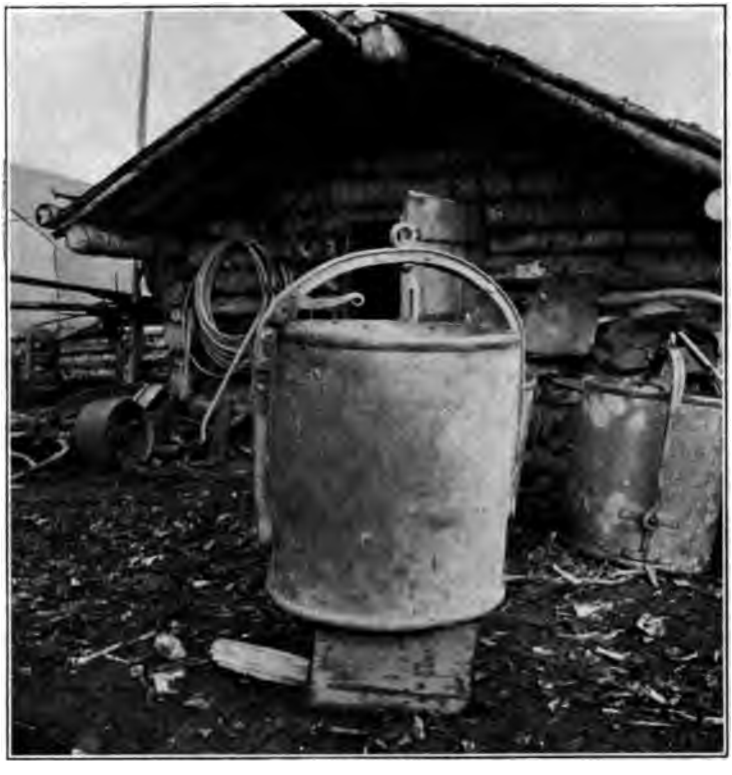
[a] The Dawson self-dumping carrier and its use is described under the heading "Drift mining," p. 94.

*A.* SKIDS FOR DERRICKING.



*B.* DERRICK BUCKET.

carrier plant of daily capacity not exceeding 200 yards. The first cost of all, including a 30-horsepower boiler, was said to be $4,500. Here the gravel is shoveled into buckets holding 8 cubic feet. These buckets are trammed on small trucks running on wooden tracks, hooked onto by the derrick cable and lifted and conveyed at same time to the sluice. The derrick boom has a radius of reach of 40 feet, and a much smaller proportion of time is consumed by the shovelers in tramming than when the fixed bucket on cable is used.[a] The expense was increased in this plant by the necessity of continually thawing with 24 points. Pl. VII, *A* and *B* (p. 64), shows the operation of the derricking plant.

As against the 5 operations of the cable tram system, there are only 3 in the derricking system if properly handled. The proof that the derricking system is superior to the wheeling and cable tram system is evident from the comparative cost. The services of a man at the dump box are generally necessary in all cases under the present sluicing practice, so that the self-dumping arrangement of the bucket helps but little.

It should be noted that the Fairbanks operator was working under higher prices generally than the Klondike miners, both for labor (at $10 a day, as against $7.50 a day) and for general supplies, and he also had an excessive amount of steam thawing. His lower cost depended primarily on his increased capacity, resulting for the most part from the increased duty of each shoveler, owing to the expeditious and adaptable system of hoisting and conveying afforded by the derrick. A derricking system in use on Seward Peninsula will be separately described.

The conclusion is that while the cable tram system, using the Dawson carrier, is excellent for drifting work, it is not to be recommended for open-cut work where it is possible to obtain space for installing a derrick.

## DERRICKING.

The plant described below is located on Ophir Creek, in the Council mining district of Seward Peninsula. The stream has been turned aside and work is proceeding in the old bed. The distinctive feature of the plant is the use of derricks in overcoming exceptional difficulties encountered in the character of the deposit. Hand labor is used in excavation, while transport of material to be washed and disposal of tailings are accomplished by derricks.

The pit in which work is being done depends for its shape and size upon the method of working and length of the derrick boom. An area of 30 feet beyond the end of the derrick boom is worked. A pit, roughly circular, having a diameter of 140 feet, is the result, since the boom reaches approximately 40 feet in its sweep, and the buckets are hauled 30 feet from the bank. Under a stratum of sand and soil, varying from 4 to 5 feet in thickness, the gravel, in places considerably mixed with sand and clay, descends to bed rock, usually 30 feet below the surface. The gravel is entirely unfrozen, rounded, and for the most part small, not over 10 per cent exceeding 6 inches in diameter, while no bowlders are found above 18 inches. Bed rock is a massive limestone, extremely irregular in its position, and causes the greatest difficulty in the extraction of much of the gold, though this same feature must have played an important part in the enrichment of the claim. (See Pl. VIII, *A*, p. 66.) In places deep holes have been made by action of water, and the depth to which work is carried in the recovery of the gold depends entirely upon the economy with which it is extracted. There is no doubt that much gold remains below, to obtain which would not be profitable under present conditions.

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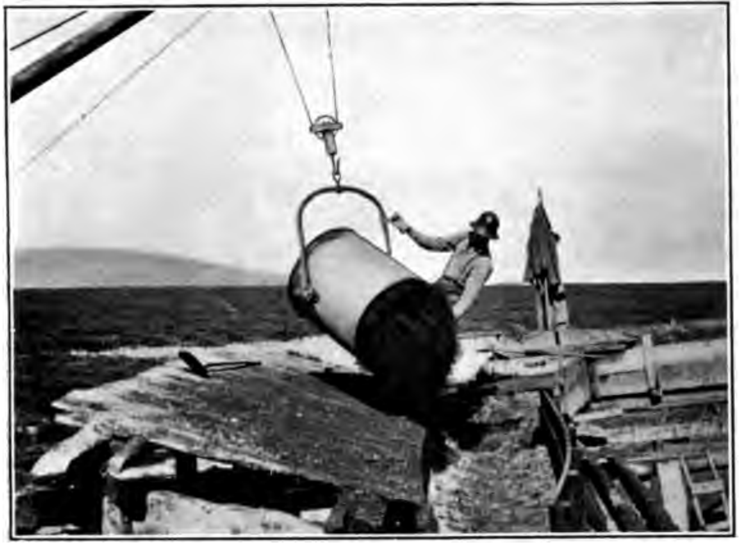
[a] See also special description of derricking, pp. 67-71.

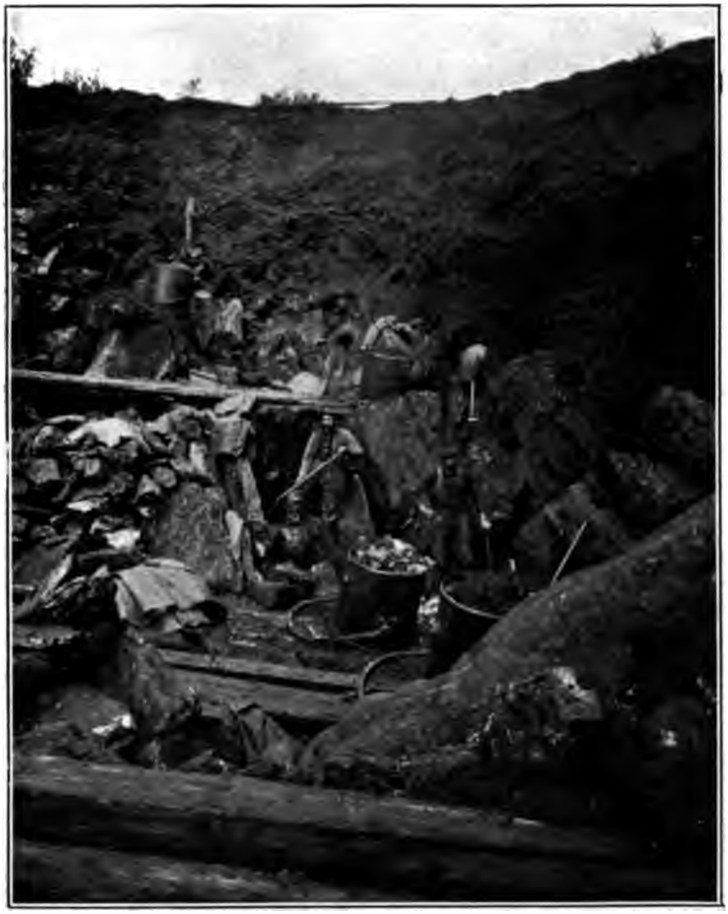
Excavation is accomplished entirely by pick, the gravel being shoveled into the derrick buckets. These buckets are hauled by the derrick line, guided by hand, upon wooden skids (see Pl. VIII, *A*), to a point directly beneath the end of the boom, where they are hoisted and carried to the dump box.

The buckets are of 11 cubic feet capacity, and are so practical as to warrant detailed description. They are made of crude oil drums or gasoline tanks, cut to a height of 2 feet 8 inches, and are 2 feet 5 inches across the top. (See Pl. VIII, *B*.) Two lugs to hold the bale are set opposite each other one-fourth distance up from the bottom. The bale is made from the original hoops of the drum. The bottom edge is strengthened by the original flange of the tank, while on the upper edge has been riveted the flange originally at the top of the tank. The bale is supplied with a catch which, when the bucket is traveling, rests in a notch constructed on its edge, which holds the bucket in an upright position. On reaching the dump box, a man on the platform with his shovel frees the catch, and the bucket dumps in turning bottom upward. (See Pl. IX, *A*.) In fitting the lugs holding the bale, a piece of iron 9 inches square is riveted to the inside of the material composing the drum, which is one-eighth of an inch thick. To the outside of the drum a strip 9 inches long, 2½ inches wide, and one-half inch thick is also riveted, and to this the lug (2 inches long) is welded, making a very strong construction. These buckets weigh 140 pounds and, including labor, blacksmith fuel, and original price of drum, cost about $25.

The skids upon which the buckets are hauled are ordinary smoothed timbers, and are not fastened down. This admits of rapid change in accommodating the hauls to the advancing work. It is desirable to have these timbers as much as possible in a radius of the circle *described by* the derrick, as this gives a straight haul on the boom.



*A*. DUMPING BUCKET.



*B*. ROUGH BED ROCK AND BOWLDERS PILED IN PIT.

The derrick in use at this plant is of a very simple and practical design, there being but one haulage line, which with gravity utilized in the lean of the derrick accomplishes the desired work. The derrick leans toward the pit in which excavation is taking place—i. e., when at rest the boom swings away from the hoisting engine. (See Pl. X.) The hauling line passes through a block at the foot of the mast and a few feet in front of it, or toward the direction of haul. In this position immediately upon receiving a hauling strain the boom tends to move toward the dump box, and once there, having disposed of its load, returns by gravity to the pit. To the end of the boom is attached a rope by which a man in the pit giving his entire attention to this operation can regulate the swing of the boom to a nicety. By snubbing this line about a post set firmly in the ground, hauls can be made in any direction with no inconvenience from the swinging of the boom, and the dumping of the load on the platform can be accurately adjusted. The derrick on an average lift raises material about 25 feet and carries it approximately 85 feet to the sluices. The amount of material handled is generally 500 cubic yards in twenty-two hours. Power is obtained from a 15-horsepower boiler and 8-horsepower double engine hoist, which burns one-half cord of wood in twenty-four hours at $15 a cord. The derricks are of Washington fir, both boom and mast being 40 feet long. The hauling line, one-half inch in diameter, is of crucible steel wire, plow-share steel not bending easily enough, and the guy lines, three-eighths inch in diameter, also of steel, are tightened by watch tackles.

Stones over 12 inches in diameter are not sent through the sluices, being piled by hand in the center of the pit. Larger stones and bowlders of bed rock are handled by the derrick and piled in the same manner. (See Pl. IX, *B*, p. 68.) The removal, however, of stones exceeding 2 tons in weight is not attempted with the derrick. A move is necessary each thirty days, and in advancing the entire plant approximately 110 feet eighteen hours are generally consumed. An auxiliary flume and dump box must of necessity be used to permit such rapid change.

The dump box, which is furnished with a platform sufficiently large to prevent spilling of material, is from 6 to 12 feet above the ground. It is 16 feet long and has a grade of 14 inches. Below it are 132 feet of boxes, 20 inches deep and 16 inches wide, with a grade of 7 inches to the box length (12 feet). These are furnished with pole riffles made 2 inches square, capped with iron, and will last two seasons. At the end of the string is a very efficient undercurrent, invented on the ground. It consists of an ordinary sluice box furnished with iron bars approximately three-eighths or one-half of an inch in diameter, running lengthwise with the box and placed side by side. Beneath these bars, held by Hungarian riffles, is a quantity of quicksilver, which catches any gold that may have passed the preceding string. Such an arrangement, though of considerable use when gold is generally coarse, will not serve when fine gold is encountered. An undercurrent of greater surficial area and greater grade, carrying the water in a thin sheet, would then be demanded.

In washing the gravel 150 miner's inches are used. The water is brought from Ophir Creek in a ditch 1½ miles long. It is 6 feet on the bottom, 9 feet on top, and has a grade of one-tenth inch to 100 feet. It cost $8,000.

Tailings are handled by a derrick and a self-dumping scraper. Power is supplied to the latter by hauling directly from the derrick boom, and its operation is directed by two men. In this case the derrick leans toward the dump pile, to which it carries its load by gravity, being hauled back by the steam winch. To the scraper is attached a rope which, on tightening as the derrick swings, dumps the load at its destination. (See Pl. X.) Both the derrick boom and mast are 40 feet long and are arranged like the excavation derrick, except that the mast leans in the opposite direction. A 15-horsepower boiler and 8-horsepower winch are used, and there are three men on each shift. The tailings derrick is not occupied more than one-fourth the working time of the pit derrick.

In all from 55 to 60 men are employed about the plant. A man is continually needed at the mud box to trip the bucket and to feed the gravel as regularly as possible to the sluices from the platform. A winch man is needed at each derrick. A man in the pit devotes his entire attention to the snubbing line, while two men are necessary on the scraper. The latter feature could possibly be improved by some mechanical method not requiring the hand labor. Wages are 50 cents an hour, with board, and work is continued eleven hours each shift.

Besides the derrick plant above described another, seen on Ophir Creek, made use of iron skips of 1¼ cubic yard capacity, which were run on trucks to the working face and, after loading, were trammed within reach of the derrick and lifted to the sluice. As nearly as could be learned, while the capacity of this plant was from 15 to 20 per cent higher than the one previously described the work was not being done so cheaply. In the last case it was necessary to pump the seepage water from the pit, while in the first case all seepage water was naturally disposed of by draining into the peculiar cavernous limestone bed rock. In derricking plants in general, large rather than small buckets or skips are to be recommended, but the various elements of the plant should be so coordinated that the capacity of the derrick is not above that of the shoveling and tramming or of the sluice.

The derricking plant seen on Pedro Creek, in the Fairbanks district, has been described in connection with the method of using the cable

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tram and self-dumping bucket in open-cut work. Its efficiency was found to be high and its cost low in comparison with this well-known Klondike method. In general it may be said that derricking is a simple, efficient, adaptable, and comparatively cheap method of working open cuts where gravel must be shoveled into the first receptacle by hand and the bed rock cleaned by men.

### TRACKS AND INCLINES.

The track and incline working at No. 8, Anvil Creek, Nome district, is fairly representative of open-cut work by this system. (See fig. 9.) A special feature of the plant, however, is the removal of the overburden, in all 7 feet of muck and 5 feet of barren gravel, by hydraulicking. Seven hundred miner's inches are used through a giant, under a pressure of 200 feet obtained by pumping. After the barren ground has been removed the remaining gravel, averaging about 6 feet in depth, is shoveled into cars, pushed to the foot of an incline, elevated, and washed. The tailings are impounded on the claim, being scraped from the end of the sluices by 2-horse scrapers.

At the time of observation 6 cars were in use, though this number can be varied to suit the conditions in hand. Two men are employed with each car. They take turns in shoving the car to the incline, one remaining at the bank with pick. The cars at this plant were of a capacity of 22½ cubic feet, though larger ones might be found more economical. From 250 to 300 cars are run up each 10-hour shift.

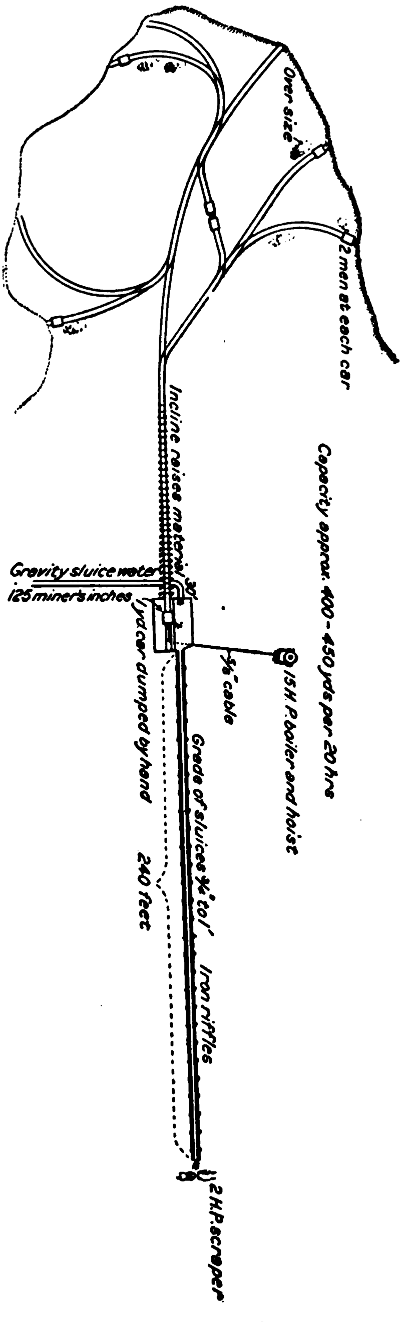


FIG. 9.—Plan of track-and-incline system, Anvil Creek.

*The material is lifted* 30 feet to the mud box and there dumped by hand. A man is continually employed for this duty. He also watches large bowlders and prevents jamming in the flume. However, very few large stones are allowed to pass through, for in the work of excavation the larger rocks are piled at one side of the track.

One man is employed at the hoisting winch and does his own firing. Three 114-pound sacks of coal are consumed each shift. A 15-horsepower boiler is used with vertical engine, and a five-eighths inch cable hauls the cars.

The dump box is 24 feet long, 4 feet wide, and 3 feet high, and has a grade of 1 inch to 1 foot. Following this are 240 feet of boxes with a grade of three-fourth inches to 1 foot. The boxes are 16 inches wide and 14 inches high, and are supplied with cast-iron grate riffles. These are 16 inches square and can be used either as Hungarian or longitudinal riffles. In washing, 125 miner's inches are used, all obtained by gravity from a claim above. After being used the water is caught in a retaining dam, settled, and carrried by flume to a lower claim, where it is sold. Two 2-horse scrapers are used to keep tailings clear, and all gravel is impounded by a brush dam upon the claim.

A little gold is caught in the entire string, though by far the greater proportion is retained in the mud box and that following. In the mud box a perforated iron sheet laid over the riffles aids the moving of material and prevents clogging. In the center of this sheet is placed a heavy iron plate which receives the heavy fall from the cars and saves the bottom of the mud box.

Bed rock on this claim is mica-schist, though in places the pay is underlain by a stiff clay. The gold sinks in the broken schist about 2 feet, but not at all in the clay.

A bed-rock drain, 250 feet long, has been cut nearly level, which is joined by small drains from various parts of the pit. Only about 15 miner's inches of water were running from the drain at the time of observation. Men are paid $5 a day, with board.

The advantages of the car-and-incline system are not especially pronounced. The method of stripping used in the above-described plant, or at any plant where water under pressure is available can hardly be improved upon. Where the overburden is thawed or where frost is encountered only in spots, as in the claim under discussion, hydraulicking has been found the most economical and efficient means of removing material. The car system, however, has little of solid economic advantage to recommend it. In order to increase capacity more men must be employed to shovel. In this system an undue proportion of time is consumed in tramming the cars to the top of the incline. A mechanical system, such as derricking or steam scraping, has many advantages over the track-and-incline. In the Klondike, so far as could be learned, the track-and-incline system does not make so economical a showing as other methods of working open cut, although the opportunity for comparison of efficiency was limited.

## THE STEAM SHOVEL.

Where the ground exceeds 12 feet in depth and is unfrozen, certain conditions justify the installation of a mechanical excavator of large capacity, say 1,000 cubic yards or more in twenty-four hours. It will seldom be found advisable to install so large and so cumbersome a machine as the steam shovel in the creek diggings of Alaska, for the ground is rarely deep enough to justify the expense of installation. On bench diggings, however, where the pit can be drained by gravity, the steam shovel has a value which has probably been underestimated. The fact that water under pressure is difficult or impossible to obtain for the hydraulicking of benches raises the question whether these bench gravels can not be excavated by other means. The value of the steam shovel lies in the fact that it performs for the earth worker that portion of his work which would otherwise be most expensive.

The ground which the shovel is to move must possess certain favorable conditions. In the first place, it must be entirely free from permanent frost when the dipper lip of the excavator attacks it. If the ground holds a certain amount of permanent frost and this can be thawed by ground-sluicing the muck off at a period far enough ahead, the shovel may still have a profitable field, but its operations are likely to be more expensive. Heavy gravel and bowlders are easily handled by the mechanical excavator. It is safe to say that no quality or state of the Alaska gravels makes them unfit to be dug by the dipper except the frozen condition.

The bed rock must be of sufficient softness to allow the dipper lip to dig far enough into it to recover all the gold; otherwise a gang of men will have to follow the shovel to clean bed rock, and a large part of the value of the shovel will be lost. In one attempt to operate with a steam shovel in the interior of Alaska it was found that as many men had to be employed in cleaning the bed rock as in the entire remainder of the plant.

A prime essential to success, as has been proved by experience, is that the washing plant shall be isolated, the gravel being conveyed from the dipper to the sluice by some form of tramming. If cars are used, they should be large—2 yards capacity, or even larger. Under ideal conditions the dipper will dump into cars which run by gravity to the hopper of the washing plant, the water being brought to the sluice also by gravity. The full cars would in this case carry the empties back to the pit. It is rarely possible, however, to find an auriferous gravel deposit in which such eminently fit conditions exist for work. The tramming, even when it must be up an incline to a height of 35 feet above the pit floor, adds proportionately little to the expense, as will be seen by figures following. It is a common fault in all steam-shovel operations that the shovel is ahead of the car discharge.

A partly idle steam shovel, however, is not so serious a fault as idle men, since the shovel draws no pay.

As to capacity, it is likely that a 2-yard shovel, fitted with extra long boom and 1¼-yard dipper, will be found most economical. A 25-foot bank can be dug and caved. If the sluice and tramming capacity is 700 cubic yards a day, the shovel will easily supply the material if no frost is encountered.

Pl. XI, *A*, shows one of the steam shovel plants in Alaska, operating on Anvil Creek. The work done by this shovel is considered satisfactory, though its installation is experimental, and a larger one is planned. It supplanted a hydraulic-elevator system which was condemned. The 25-ton shovel (three-fourths-yard dipper) is working on an 18-foot face. It has not reached the bottom of the gravel, and must make another cut 7 feet lower vertically before all the pay is extracted. It is said that from July 23 to September 1, 1904, 25,000 cars of 1¼ cubic yards capacity were dug and moved to the sluice boxes at a working cost of 12 cents a yard. The low bench is sufficiently above the level of the present creek to permit the pit to be drained, the seepage water being handled by bed-rock drain. The workings were visited a second time three weeks after the present cut was made. A complete section of the bank to bed rock is given below:

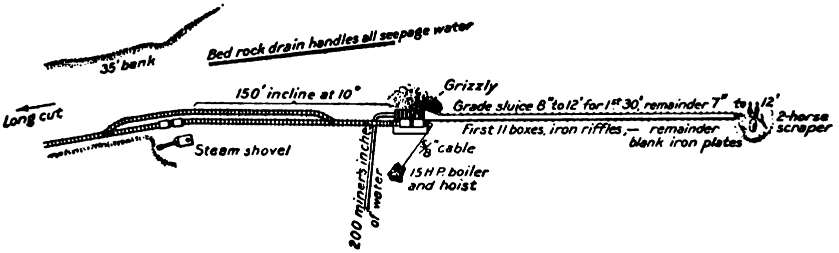

35' bank
Bed rock drain handles all seepage water
150' incline at 10°
Grizzly
Grade sluice 8" to 12' for 1st 30', remainder 7" to 12'
Long cut
2-horse scraper
First 11 boxes, iron riffles,— remainder blank iron plates
Steam shovel
⅝" cable
15 H.P. boiler and hoist
200 miner's inches of water

FIG. 10.—Plan of steam-shovel operations, Anvil Creek.

| | Feet. |
|---|---|
| Muck | 3 |
| Fine gravel and sand | 5 |
| Fine subangular gravel | 5 |
| Large subangular schist and limestone fragments, stained with iron oxide, with a few bowlders up to 3 feet in diameter | 15 |

The upper 3 feet of muck was ground-sluiced; the remainder was moved by the steam shovel.

This frozen ground illustrates the peculiarly trying conditions with which the Alaska placer miner has frequently to deal. None of the ground encountered in 1904 was frozen. In the early part of the season of 1903, however, on account of the light snowfall during the preceding winter, the sides of a cut operated on were frozen to a depth of 8 feet and to a distance of 15 feet into the bank. This was annual

*A.* STEAM SHOVEL, ANVIL CREEK.

*B.* STEAM SHOVEL, BEAR CREEK, KLONDIKE.

and not permanent frost. Had the attempt been made to work the ground at that time with the shovel the operations would have been greatly delayed. In Pl. XI, *A*, one of the frequent defects of the steam-shovel system is apparent—the shovel idle and waiting for cars. In this plant, after the shovel had worked to a considerable distance from the bottom of the incline, the 1¼-yard cars, 3 in number, were trammed to the bottom of the incline by horses and hoisted to sluice by a 15-horsepower hoist. The tramming may be more cheaply accomplished, where there are several years' work ahead, by a small locomotive in the pit running to the bottom of the incline or, under favorable conditions, directly to the washing plant. Fig. 10 shows a plan of the operation under discussion.

The tramming system illustrated by fig. 11, used at Galesburg, Ill., in one of the shale pits of the Purington Paving Brick Company, is considered highly economical in steam-shovel work. As may be seen, the locomotive occupies a position intermediate between the two trains of cars, which deliver two ways to the bottom of two inclines leading to the hoppers of the clay machines at each end of the pit. In placer operations, if conditions admit of dividing the water to two washing plants, this system is to be recommended, as it allows of rapid delivery of the cars from the shovel. In this plant the locomotive keeps 20 cars going, each of 2 yards capacity, tramming them alternately, in trams of 6 and 4, two ways to the ends of the pit, whence they are hauled, two at a time, to the hoppers. When empty they run down and are switched automatically to the empty tracks. The 90-ton shovel, of 5-yard dipper capacity but fitted with 2-yard dipper, is ahead of the capacity of the clay machines, but must be used owing to the difficulty of digging the firm shale which composes the bank. The actual yardage moved, working nine hours a day, is 670, or at the rate of 1,488 yards in

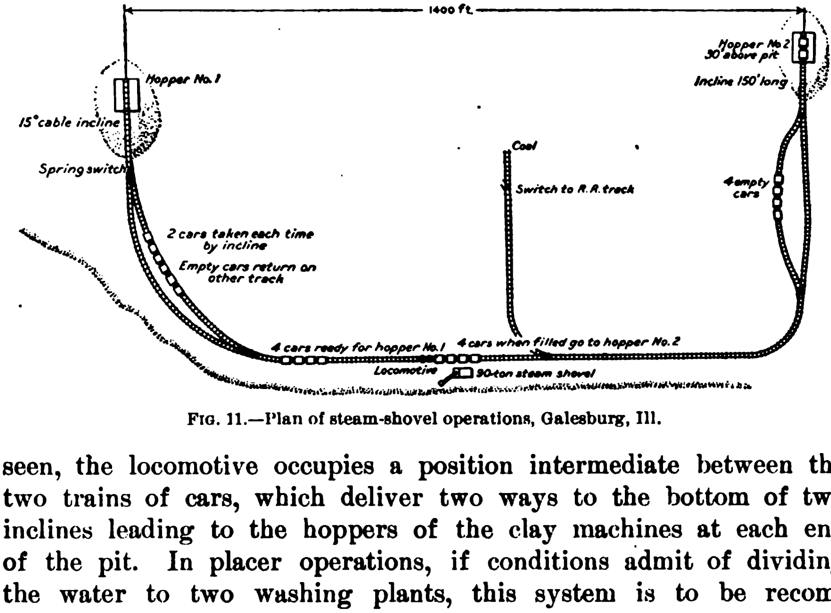


FIG. 11.—Plan of steam-shovel operations, Galesburg, Ill.

twenty hours. It is estimated that this is less than one-half the amount that could be handled were the shovel digging loose gravel. As it now works, the shovel is digging only one-third the time.

For moving up the cars within reach of the dipper as each one is filled the device shown in fig. 12 is used. This was devised by Mr. W. S. Purington, and has been in successful use at this plant for five years. The long cylinder, made with casting to attach to the shovel, here shown on the near side, contains a piston of equal length, which is supported on suspended track and wheel as it leaves the end of the cylinder. To the near end of the piston a cable passing over a sheave is attached. The other end of the cable is hooked to the corner of the gravel car, steam is turned into the near end of the cylinder, and as the piston travels back toward the forward end of the shovel-car the gravel car is hauled by the cable an equal distance, from 5 to 7 feet, as may be required. Steam is then turned into the cylinder, allowing the piston to return and the cable to free itself; the cable is unhooked and pulled by the car man to the following car and hooked in readiness to pull it along. The amount of steam and the time consumed in the operation are so small as to be almost negligible. By passing the cable around the body of the shovel car over a second sheave the cars on the opposite side of the shovel can be moved, when the relative position of the shovel is reversed.

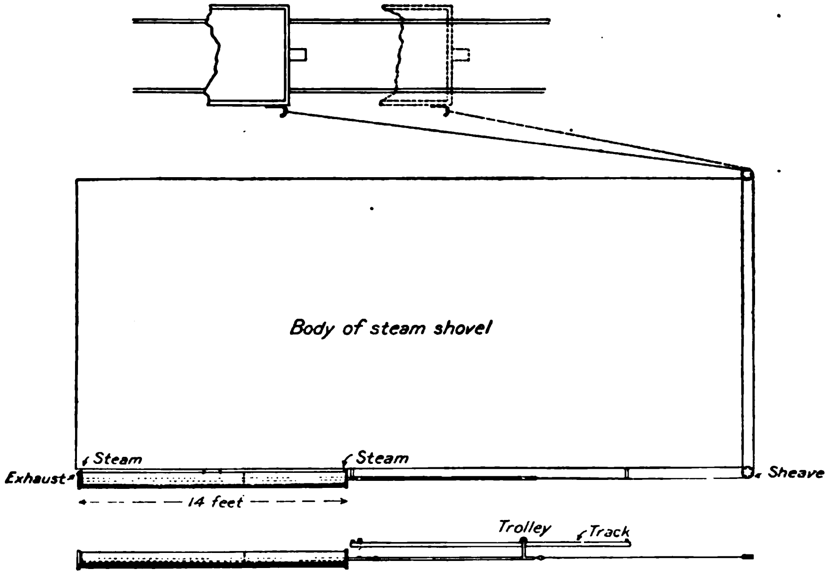


FIG. 12.—Pulling-up device for steam shovel.

Dipper chains are generally preferable to cables in placer operations, as illustrated by the experience of the shovel dredge in Solomon

River, elsewhere described. A link in a chain can be repaired where a wire cable can not.

The moving of the shovel as the digging progresses must be brought down to the most economical and rapid system possible. In the Galesburg operations wooden skids of 6-inch timber, 3 by 3 inches in dimensions, are hung from the jack frames by chains, and thus require no attention when the shovel is moved up. On these skids rest 6-inch blocks, 2 by 2 feet, on which the jack shoes bear.

In the Anvil Creek plant the present arrangement for dumping the cars at the sluice is unsatisfactory, and will be changed. Self-tripping two-way dumping cars will be substituted for the man who trips the cars. The sluice presents no new features. It is 500 feet long by 32 inches wide, by 36 inches high at the dump box, and narrows to 26 inches wide by 24 inches high, in 10-inch grade, and is furnished with angle iron riffles. The cost of digging, tramming, and dumping the gravel to sluice is said to be 12 cents per cubic yard.

The cost of steam-shovel operations in Alaska is not evident from the cost sheet (table 1, p. 38), as the figures there given cover the sluicing, often with purchased water, stripping, amortization, etc. In the Anvil Creek operation the actual working cost of digging the gravel, tramming and hoisting, and tripping the cars to the sluice box is given below, as estimated by the superintendent, the figures covering 810 yards in twenty-four hours, and including superintendence on the ground, labor, crude oil fuel at $3 a barrel, lubricants, etc.

*Cost of steam-shovel work at Anvil Creek, Alaska, per cubic yard.*

| | |
|---|---|
| Digging | $0.045 |
| Tramming to incline | .025 |
| Hoisting | .018 |
| Dumping | .011 |
| Proportionate superintendence on ground, and incidentals | .021 |
| Cost per cubic yard | .120 |

Figures showing the cost at the Galesburg plant, above referred to, are available in detail, for handling 17,422 cubic yards of shale per month of 26 nine-hour days, digging from 50-foot bank with a shovel fitted with 2-yard dipper, delivering to 20 two-yard cars trammed 2 ways, 1,500 and 2,000 feet, respectively, to bottom of 2 inclines, hoisted by cable to hoppers at elevation of 20 feet above the track and dumped to hoppers.

*Cost per month of steam-shovel work at Galesburg, Ill.*

| | |
|---|---|
| 1 engineer, shovel | $110.00 |
| 1 crane man | 85.00 |
| 1 fireman, at 22½ cents per hour | 52.65 |
| 3 trackmen, 17½ cents per hour (shovel) | 128.85 |
| 1 locomotive engineer | 80.00 |

*Cost per month of steam-shovel work at Galesburg, Ill.*—Continued.

| | |
|---|---|
| 1 switchman, at 20 cents per hour | $46.80 |
| 2 hoist men, at 20 cents per hour (hoist men dump the cars) | 93.60 |
| 1½ tons coal for shovel, at $2 | 78.00 |
| ½ ton coal for locomotive | 26.00 |
| 1 ton coal for 2 hoists | 52.00 |
| Total | 752.90 |
| Cost per cubic yard | .0432 |

Charges for superintendence, oil, waste, and incidentals are omitted from the above. Including these the cost is practically 5 cents a yard. The comparison below, however, is based on the figure of $0.0432.

If two of the trackmen be charged to the shovel, the digging operation and transfer from bank to car costs $0.0236, or 54.6 per cent of the operation considered. In other words, the digging here costs little more than the long tram, hoisting, and dumping, and thus the object of using mechanical appliances is attained. A striking comparison is afforded by the cost of operating a placer which uses the car-and-incline system and the labor of men to fill the cars. If the duty of a man shoveling gravel into a car is taken at 12 cubic yards in nine hours, and the men are paid the lowest wage given in the above tables, namely $42.85 per month, 56 men, say, would be employed filling the cars, and the monthly cost would be $2,305, or $0.132 per cubic yard, 5.6 times as great as that with the shovel, or 84.8 per cent of the entire operation considered. In Alaska where wages for shovelers are from four to six times as great, the percentage of cost would be much greater.

Although the cost of 12 cents on Anvil Creek appears attractively low, it must be remembered that it does not take into account the labor of shovelers cleaning bed rock after the shovel. This operation, unfortunately, can rarely be dispensed with, at least in portions of the ground, and it will naturally increase the cost of getting the auriferous material into the cars. In some places in Alaska the slabs and leaves of bed rock must be scraped with brooms in order to recover all the gold. The prospective steam-shovel miner should keep in mind the fact that a machine which will handle a large quantity of material, however cheaply, is of no avail unless it can succeed in extracting a large percentage of the values.

The steam-shovel plant at the junction of Bear Creek and Klondike River occupies the flood plain of the latter stream, a fact that influences greatly the economical excavation and disposal of material, since both lack of grade for washing gravel and dump for disposal of tailings must be supplied artificially.

This steam shovel (see Pl. XI, *B*) digs in a pit 20 feet below the surface of the flat. The dipper empties into cars which are pushed by hand to the foot of an incline and raised by steam winch to the platform of the washing frame, where they are dumped by hand into a hopper. The material passes through a trommel, and the oversize falls into a self-dumping carrier and is elevated to dump. The undersize passes over tables and through sluices, and the fines are also raised by steam, a scraper being employed to secure dumping ground. There are, therefore, four distinct elevations, viz, by the shovel, the gravel by the tram, the coarse tailings by the bucket, and the fines by the steam scraper.

The pit in which the shovel works varies in depth from 20 to 24 feet. The gravel lies beneath 3 feet of muck, is generally mixed with considerable sand, and extends to bed rock. The material is well rounded and contains very few stones (but 1 per cent) that are over 18 inches in diameter. Considerable black sand is found in the clean-up. In spots the ground is frozen, necessitating the occasional use of steam points. Steam is conveyed across the cut in a covered steam pipe hung from a stretched cable. A schist bed rock, generally hard, is excavated for about 2 feet by the shovel.

The shovel weighs 35 tons and revolves 360° upon a turntable. The dipper holds 1 cubic yard, and the machine has a capacity of 1,000 yards in ten hours, though this can not be obtained owing to inadequate tramming facilities. Its cost on the ground is about double that charged by the manufacturer. The boom, when horizontal, reaches 22 feet beyond the bow, but when at level of track only about 14 feet. Three-quarter-inch cables are used in transmitting power. The faults of cables in steam-shovel practice in inaccessible localities have been commented on above. The dipper contains no new features. Prongs are added to save lip, the central one being longer than those adjacent. The machine has 3 engines and uses in all about 40 horsepower.

For conveying material three cars of 2 yards capacity each are used, running on a 3-foot gage track with 16-pound rails. The cars are pushed by hand to the foot of an incline, which has an angle of 22° and extends 100 feet, and thence elevated by cable connected to a 40-horsepower hoist. Two hundred and fifty cars are usually raised in ten hours. On reaching the platform the cars are dumped by hand and returned by gravity to the foot of the incline.

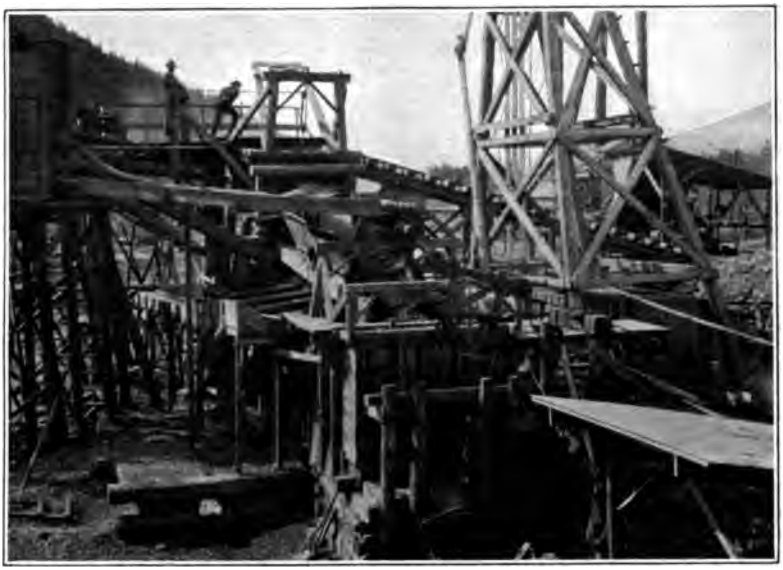
From the hopper into which the cars dump the gravel is fed into a revolving trommel 16 feet long and 3 feet in diameter supplied with a worm upon the inner surface and punched with holes varying from one-half to 1½ inches in size. The rate of feed is regulated by hand, a gate being raised and lowered when necessary. Water is supplied to the trommel by a 6-inch longitudinal pipe. All oversize passes directly over a sheet-iron chute into a square hopper, from which it falls by gravity into a self-dumping carrier and is elevated approximately 40 feet upon a dump. A 10-horsepower winch does this duty. The hopper is furnished with a heavy iron gate, raised by the steam winch running the self-dumping carrier, and is lowered by its own weight. The fines (the material passing through the trommel) fall directly on gold-saving tables similar in design to those used on many gold dredges. These tables (see Pl. XII, *A*), which are fed by two 4-inch pipes, are six in number (three on each side of the trommel) and have a gold-saving surface of 80 square feet. They are fitted with expanded metal and cocoa-matting riffles, which prove to be very efficient, the greater part of the gold being caught near the upper end of the first table. (See Pl. XII, *B*.) The material then passes to a longitudinal sluice which consists of eight boxes 22 inches wide furnished with Hungarian riffles constructed of 1- by 3-inch inclined cleats capped with three-sixteenths-inch iron. The tailings are removed by a steam scraper and elevated upon a dump. The winch for the scraper is 30 horsepower and elevates 400 4-wheelbarrow units 15 feet in ten hours, seven-eighths-inch cable being used. It will be seen that the above double elevation of material must be costly, using, as it does, not only much power, but also additional men on separate winches.

The greater part of the water for sluicing, 300 inches, is brought by flume from Bear Creek. The remainder, amounting to 100 inches, is raised from the pit by a series of pumps. A No. 6 centrifugal pump raises the water to the foot of the washing apparatus, from which it is lifted 25 feet by a No. 8 centrifugal and used for sluicing. A 35-horsepower engine does the latter work. An additional pulsometer pump is occasionally used in the pit.

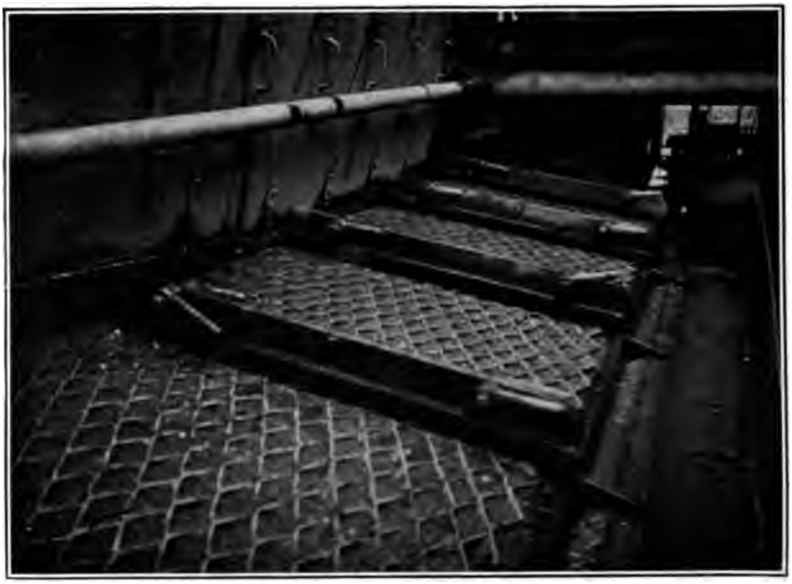
The power plant contains two 60-horsepower boilers and one 50-horsepower boiler, wood being used for fuel. Sixty men are employed about the plant, the daily wage of the laborer being $4 and board. The men work two 10-hour shifts throughout the working season.

The above-described plant will undergo considerable change in the near future, improvements designed to effect greater economy being contemplated. It is planned to haul all material excavated a distance of 2,000 feet by locomotive, in trains of 6 cars, to a washing plant on the banks of Klondike River, where the ground is 8 feet below the upper edge of the pit, and then wash it by gravity water from Bear Creek. High water in the Klondike will greatly economize the handling of tailings, and it is hoped the full capacity of the shovel may be utilized.

A machine run on principles radically different from those described above is illustrated by a steam shovel on Eldorado Creek, in the Klondike region. In this case the use of a tramming system is entirely avoided by so arranging the plant as to allow direct dumping by the dipper into the sluice boxes. In this case the shallowness of the ground and the use of a 50-foot boom permit such an arrangement. The work lies in the creek bed proper and cuts are made longitudi-



*A*. STATIONARY WASHING PLANT, KLONDIKE.



*B*. COCOA MATTING AND EXPANDED METAL RIFFLE.

nally along the stream, the sluice boxes being moved every three or four days, following the progress of the shovel. This practice alone involves both good and bad features. The practice of dumping directly into the sluice is, theoretically, much to be desired, embodying, as it does, only one elevation of the gravel, an arrangement which consumes the least power and time—two all-important factors. The moving of the sluice boxes, however, in this case consumes much time—from one to two days for each move—and, wages being $6 a day and the season very short, constitutes a serious drawback, greatly reducing the capacity of the plant. The boom of the machine, as stated, is 50 feet long and hoists the gravels 30 feet above the cut. If an arrangement facilitating the moving of the boxes could be devised, so as to avoid the long delays caused by this operation, a very satisfactory result might be effected. Such a system requires a relatively shallow bank, however, and it is doubtful whether it is available for steam-shovel operations in general.

In this instance the gravel to be taken out is frozen, and continuous thawing must accompany the advancing work. In this work 15 steam points are driven into the bank to bed rock, where they remain twenty-four hours, the process consuming during that time 3 cords of wood. A 25-horsepower boiler is used for this duty. (See Pl. XIII, *B.*)

Water is very scarce at this plant and is used repeatedly, being raised from the pit by a centrifugal pump and carried back to the sluice boxes in a flume. The pump also serves to drain the pit. The sluice boxes have a grade of 9 inches to 12 feet, but, with the water at hand, are unable to keep pace with the capacity of the machine.

The shovel carries a one-yard dipper operated by chains. It is claimed that 1 cubic yard is moved every two minutes. Bed rock of a slaty nature is taken up to a depth varying from 2 to 5 feet.

The problem of disposal of tailings is a difficult one and has not been solved economically. A steam scraper is used and all tailings must be scraped to one side and elevated. Not only is the expenditure for wood an item, but a man must be constantly attending the scraper while another handles the steam winch.

A tramming system which would convey the material to be washed to such a point as would admit of the disposal of tailings by gravity, even if a thousand feet or more distant, would be a more economical arrangement, saving, as it would, not only all elevation of tailings, but also all delay in moving the washing apparatus.

Ten men are employed about the plant each shift, occupied in scraping tailings, forking gravel in dump box, running shovel, attending steam points, and in cleaning bed rock at places where the dipper could not effectually work.

The shovel is moved by rollers running on skids, power being obtained from the shovel winch. The dipper is furnished with soft steel prongs which wear very rapidly.

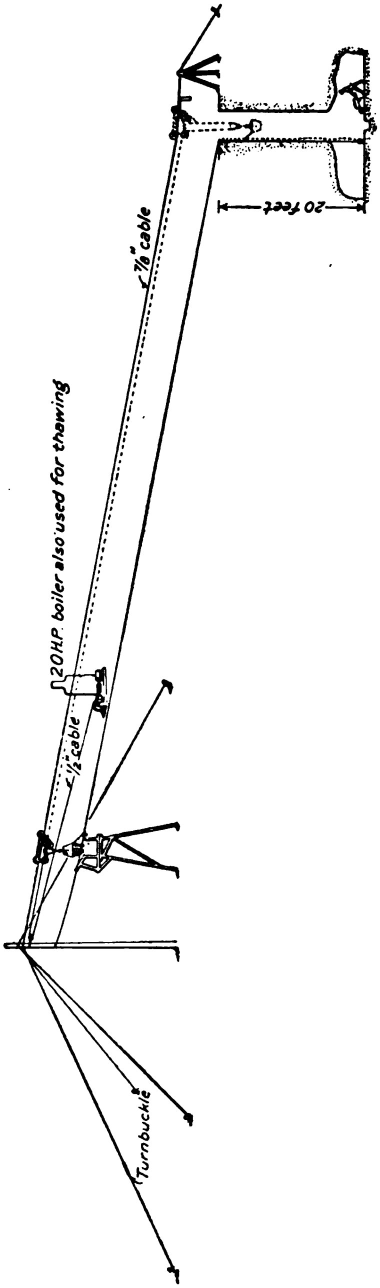


Fig. 13.—Rig for small drifting operation.

## DRIFT MINING.

### INTRODUCTION.

Northern gold-gravel deposits, that are sufficiently rich to warrant the method, can generally be exploited by drifting beneath the overburden at all times of the year. In drift mining creek deposits the gravel is hoisted through shafts, and where bench gravels are worked it is generally trammed through adits. In all portions of the interior of Alaska and in the Klondike the drifting of auriferous gravels has been carried on with fully as much activity in the winter as in the summer. As may be seen by the cost sheet, the expense of winning the gold by winter drifting and spring thawing is greater by over $1 per cubic yard than that of summer work. This increase is due to the necessity of rehauling and, in many cases, of rethawing the dumps in the separate process of sluicing. In consequence, only the richer ground, running from $6 to $10 to the cubic yard, can be profitably drifted. The price of labor during the winter in the interior is only 25 per cent less than in summer, so that this does not afford a great offset to the increase of cost from other causes. In Seward Peninsula, where the price for winter labor is only $2.50 a day and board, as against $5 in summer, it would seem that winter drifting can be undertaken more generally than heretofore. A reliable operator of Nome is of the opinion that winter drifting operations should be continued in that vicinity where timbering is necessary but thawing is not, and where the gravel can be trammed to surface through adit for $3 per cubic yard, including all expenses of extraction and sluicing-up in the spring. This is less than double the cost of drift mining in the Forest Hill divide region of California, where conditions are eminently favorable.

Neglect to sample is the worst possible practice, as many lamentable failures have resulted from overconfidence in the value of the ground taken out, and the miner who confidently waits for the big wash-up before thoroughly knowing the amount of gold in the material he is handling is frequently doomed to disappointment. The method of sampling the dump, given under the heading "Hydraulic mining" (p. 118), is to be highly recommended for all work of this nature.

Where timber is scarce and costly, and wood for fuel has to be hauled from a distance, it is advisable to consider the entire abandonment of attempts at winter work. The amount of gravel which can be hauled is less, and the cost per cubic yard is greater. The time spent in sluicing the winter dump may frequently be more profitably employed in preparing for the summer. The ordinary small rig for drifting operations without timbering is shown in fig. 13.

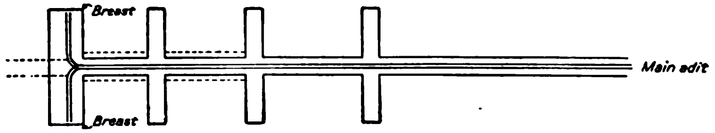


FIG. 14.—System of opening drift mine.

## DRIFTING AND TIMBERING.

Drift mining, whether the workings are approached by shaft or adit, can be most economically conducted by adopting the system which proved so successful in California. The mine is opened by a main tunnel or runway, 6 by 6 feet, which generally requires timbering, with logs 8 by 8 inches, 6 feet long, the sets having 5-foot centers. This runway is continued out to the end of the block of ground which it is proposed to work, say from 50 to 100 feet in ordinary Alaska operations. It is assumed that the runway occupies the longer dimension of the ground and approximately the center of the pay streak, say 75 or 100 feet in width. From the main tunnel drifts are run transversely each way to the outer edges of the pay. These generally do not require timbering in frozen ground. Fig. 14 illustrates the system of main runway and drifts. The gravel can now be breasted out, a face being carried the full width of the pay and the work being continued in two or more of the drifts, as desired, the breasts being always carried toward the shaft or adit mouth. The drifts may be timbered either with sets, where ground is heavy, with single posts and caps in moderately firm ground, or left entirely untimbered, as in solidly frozen ground. Fig. 15 shows the manner of timbering closely and rock filling behind when breasting out faces of loose gravel in the Hidden Treasure gravel mine in California. The sketch is made up in part from report of Mr. Ross E. Browne,[a] and in part from the writer's own observations. The system of filling behind with quartz stones is impracticable for Alaska mining; instead of this timbers should be pulled and the ground allowed to cave. This form of timbering is in use in the drift mines of Solomon Hill, Klondike, except that instead of separate base blocks sills are used. Fig. 16 shows a portion of the drift operations which were prosecuted on Solomon Hill in the early years of the Klondike exploitation. The plan of the workings is here reproduced by courtesy of Mr. George T. Coffey, manager of the Anglo-Klondike Mining Company's operations. The drifting operations in the southern portion of the ground are still in progress. Pl. XIII, *B*, shows the mouth of one of the adits leading to the operations.

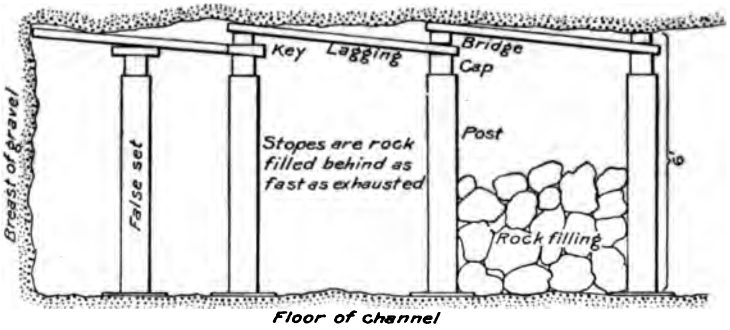


Fig. 15.—Hidden Treasure system of timbering drift mine.

The main adits are timbered with 4-inch timbers, in sets with 5½-foot centers, at a cost of $1.75 for framing and putting in per foot of tunnel run. The entire cost of driving the tunnel 5½ feet high, including steam thawing, excavating, tramming, timbering, and laying track of 12-pound rail, was $6.25 per foot. Transverse drifts were run from the main adits, and the ground was always breasted out in the direction of the adit mouth. The irregularity of the workings in a portion of the ground is accounted for by the fact that a certain amount of unsystematic work had been done previous to the purchase of the ground by the present company.

The ground is solidly frozen and had to be thawed by steam points. The rate of speed with which the work was carried on may be judged from the fact that two men, working a shift in running a straight

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[a] The ancient river beds of the Forest Hill divide: Tenth Ann. Rept. State Min. Cal., 1890, p. 452.

*A.* ADIT TO DRIFTING OPERATIONS.

*B.* THAWING AHEAD OF SHOVEL, ELDORADO CREEK.

tunnel after the points had been left in fourteen hours, averaged 5½ feet, including taking out the points, putting in timbers, keeping up the track, and driving in other points to a distance of 5 feet. The average height to which the gravel was taken out in a representative block was 4 feet. Pl. XIV, A (p. 86), shows the remnants of old drift timbers in ground which has been hydraulicked subsequently to the drifting of the rich bed-rock pay. The cost of mining was $5.50 per square yard of area worked. In drifting out the ground it was found that three points in a face 6 feet wide, allowed to thaw twelve hours and then withdrawn, the ground being "sweated" twelve hours longer, would thaw all the ground. The sluicing was conducted in the ordinary way of piling up the dump and afterwards caving and hydraulicking with a nozzle into the boxes. The actual working cost of washing was about 18 cents per square yard of area worked.

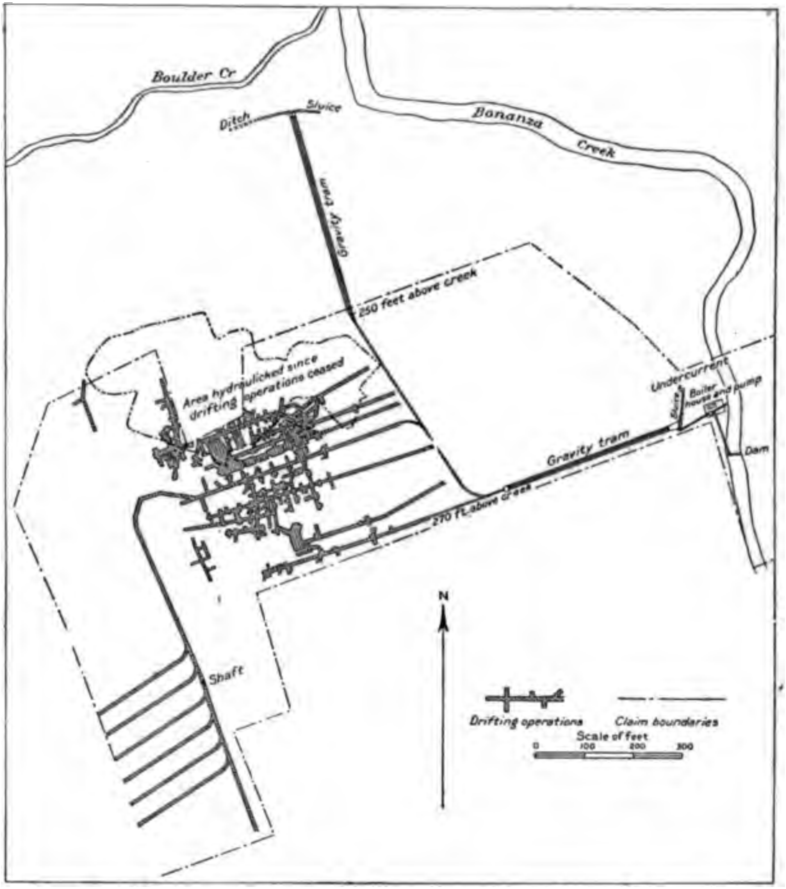

FIG. 16.—Plan of drifting operations on Solomon Hill, Klondike.

In drifting operations on a bench of Anvil Creek, Seward Peninsula, a main long adit was run 4 by 6 feet 6 inches in the clear, at a cost of $4 a foot, untimbered, for the entire width of the ground in a direction transverse to that of the creek. This was 500 feet in length and was 10 feet below the surface of the schist bed rock. The average depth of the ground was 20 feet, being 2 feet of muck, 11 feet of wash gravel containing almost no gold, and 7 feet of pay dirt, the last consisting of 5 feet of pay gravel and 2 feet of bed rock, which contained the best pay. This was the thickness drifted. The adit served as a drain for the workings. The ground was only partially frozen, and it is likely, as has been found in parts of the Alaska interior, that the constant draining of the ground assisted in a gradual thawing of the frozen parts. From the end of this main adit, a long tunnel (600 feet) was run for the length of the ground which was to be drifted, at a level of 8 feet above that of the adit floor. This was timbered with 6- by 6-inch posts, 8- by 8-inch caps, sets with 4-foot centers, lagged with 2- by 6-inch plank, top and sides. From this long drift cross drifts were run in a direction parallel to that of the main adit, at intervals of 100 feet, and with lengths of 40 feet on each side of the main longitudinal drift. The ground was then breasted, carrying a 40-foot face toward the adit. The Hidden Treasure system of timbering, using false sets, was employed. Track was laid both in the main adit and in the upper workings, and cars of 21 cubic feet capacity were employed in tramming, 4 in the main adit and from 15 to 20 in the upper workings. The stopes never exceeded 20 feet in back and forward dimensions. The timbers used were 8- by 8-inch posts, 7 feet long; 10- by 10-inch caps, 11 feet long; 5-foot centers, all ways, between posts. Boards were laid down to shovel on. The ground was so heavy it was found impossible to save the timbers, but the track was all recovered. In extracting 21,000 cubic yards, timbering cost $14,000, with lumber at $50 per thousand.

The gravel was trammed to surface and distributed on a long dump 40 feet on each side of a previously constructed 24-inch sluice 500 feet long. As usual, planks were laid on top of the sluice, and when spring opened the gravel was caved and horse scraped into the sluice. In distributing the gravel on the dump 3 men and a horse scraper had to be employed most of the time in keeping the snow off, on account of especially heavy snowfalls when the work was being done. When completed, the dump measured 700 by 80 by 20 feet, approximately. The drifting work consumed nearly eleven months, and the sluicing about two months, being delayed on account of frost in the gravel.

Labor employed averaged 45 men in two shifts of ten hours, at winter wage of $2.50 a day and board. A shift in the drifting operations consisted of 10 men shoveling into cars, averaging 8 cubic yards *to the man*; 3 to 4 trammers took care of the dirt shoveled. In sluic-

*A*. OLD DRIFT TIMBERS IN HYDRAULIC PIT, KLONDIKE.

*B*. APPLICATION OF STEAM POINT IN OPEN-CUT WORK.

ing, 10 men and two 2-horse scrapers were used to a shift. The product of the operations was $160,000; and the cost, including preliminary prospecting, was 59 per cent of the output. The work was done in 1901–2, and the opinion was given by the operator that the cost could be decreased 20 per cent under present conditions.

In drifting operations on high benches lying between Anvil and Dexter creeks near Nome the gravel must be hoisted through shafts. It is said that powder is efficient in extracting the frozen ground. This is contrary to experience in the Alaska interior, and is doubtless due to the fact that the gravel is not nearly so solidly frozen. In fact, it is said that the powder is used for only a portion of the time. From the main shaft, which must be in some cases 230 feet deep to reach the second pay streak, a drift is run the length of the pay ground. From this, transverse drifts are run at alternate intervals of 50 feet in each side of the main drift. Some operators sink the shaft on the side rather than in the center of the ancient channel, as by this method the shaft is less likely to "squeeze." The shafts are 3 by 6 feet, timbered with 2- by 12-inch plank laid up on the sides, with 2- by 4-inch timbers set vertically on the corners inside.

The face can be carried from 50 to 100 feet in length in the winter, as the constant slight freezing renders the gravel less heavy. In summer, on the other hand, the ground must be worked in small blocks. With the methods of timbering in use, 6- by 6- to 10- by 10-inch sawed posts are put in, on base block, and with a short 4- by 12-inch plank cap, posts having 3-foot centers in all directions. A space of 3 sets in width is generally carried before the ground is allowed to cave. When the ground is ready to cave a bulkhead is nailed to the second set, the timbers are pulled with ropes from the third, and the caved ground is caught by the bulkhead. The breast is then carried forward as before. No lagging is used except in the shaft and the main runway. In summer work the area of bed rock exposed in one stope rarely exceeds 10 feet square.

In sluicing the dumps from winter operations on the high Anvil benches, the expedient is adopted of building small dams of sod and erecting snow fences behind them to catch snowdrifts, so that sluicing water may be afforded in the spring from the melting snow. The remarkable phenomenon of deep-lying pay streaks on the Anvil benches has been considered by Mr. A. J. Collier[a] in his geological account of the Nome district.

On Eagle Creek, in the Birch Creek district, an interesting drifting operation was seen. It is especially difficult to work the creek claims by drifting on account of the partially thawed character of the gravel. A bed-rock drain was run in winter at one side of and parallel with the pay streak, and low enough to drain all the ground which it was proposed to work. The drain was covered and lagged with poles and thus rendered permanent. A shaft was then sunk at the lower end of the ground, 4 by 8 feet in dimensions and 20 feet deep. This was timbered only for the upper 14 feet, and was sunk in three days by 2 men. A tunnel was then run upstream the length of the ground, wide enough to admit 8-foot caps. Posts were 5 feet long, all timber being 9 inches square. Sets had 4½-foot centers, and the tunnel was lagged overhead with 2½-inch flattened poles. A set of timbers delivered cost $4, there being 34 caps to a cord of wood. Lateral drifts 8 feet wide were driven at intervals of 8 feet, 4½ feet in thickness of gravel being taken. The gravel was wheeled to the shaft and windlassed to surface, about 25 cubic yards being raised in ten hours. The lamentable lack of any tramming arrangement for transporting the gravel to a distance resulted occasionally in the caving in of the ground from the added weight of the dump. The gravel so laboriously raised was thus lost. Some idea of the difficulties of operating on Eagle Creek may be seen from the following figures: Wood was $10 a cord, not excessively high; lumber, $180 a thousand; and freight-packing rates from Circle on the Yukon, 25 cents a pound in summer and 8 cents in winter. Considering the remoteness of the district, the work carried on was remarkably good and systematic.

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[a] The placer deposits of Seward Peninsula: Bull. U. S. Geol. Survey, in preparation.

## THAWING.

One who has never visited the interior of Alaska finds it difficult to conceive the formidable condition of solidly and perpetually frozen alluvium which is there encountered by the placer miner. The quantity of water in the form of ice which occurs in the frozen gravel averages about 25 per cent; while in the overlying fine, black silt which forms the overburden the quantity of ice varies from 50 to 75 per cent. Whatever may be the physical composition of the material, it forms a mass as solid and as difficult to penetrate as solid stone, and can be disintegrated only by exposure to the sun's rays or by the long-continued application of some form of energy artificially applied.

In any open-cut operations the dense blanket of moss, from 12 to 18 inches in thickness, which covers the frozen ground, must be broken into and turned over either by adzes or plows before the action of sun and water can take any effect on the underlying muck. As has been said, this effect is remarkably rapid when the water is allowed to run over the exposed black and icy mass.

A curious condition exists in the treeless areas of Seward Peninsula. Wherever a growth of stunted willows occurs, the ground beneath is found to be thawed, and wherever the willows are replaced by moss the ground is frozen solid. The rule is not without exceptions, but has been found to have rather general application. The distribution of the willows and moss-covered patches in the creek bottoms and along the low valleys appears to be entirely irregular. For further notes on the frozen and half-frozen conditions existing in the interior reference may be made to the matter presented under the heading, "Prospecting."

Drifting operations in the creek deposits of the Klondike, Birch Creek, Fortymile, and Fairbanks districts of the interior are nearly always carried on in solidly and perpetually frozen ground. A necessary accompaniment of the work is the thawing of the ground by artificial means. Mr. Greenleaf W. Pichard has thoroughly investigated for this report the possibilities of using any form of the electric furnace for thawing the frozen gravel, and has reached the conclusion that electric thawing is impracticable. The thawing of ground which lies open to the air, by the combined agencies of sun and water, will be considered under the heading "Hydraulic mining." Artificial power, through the agency of the steam point, is only in rare cases applied in open-cut work. The Klondike district afforded a few examples of this application of power, notably in connection with dredging operations on Bonanza Creek, the steam-shovel operations on Bear Creek, and in two open cuts on Upper Dominion and Hunker creeks. Pl. XIV, *B* (p. 86), shows an application of the steam point to open-cut mining on Hunker Creek. It is difficult to determine the efficiency in open work, as, naturally, a portion of the thawing is done by the sun. From the data collected, however, it does not appear to be any greater than its underground efficiency, which will be presently discussed.

The method of thawing gravel underground by wood fires is expensive and, unless the conditions are very exceptional, is not used in those districts where transportation facilities permit the bringing in of boilers.

According to experience on Deadwood Creek, Birch Creek district, the efficiency of a wood fire in creek ground was as follows: A fire taking three-fifths cord of wood (at $12 a cord) is built against the face of the bank. The pile of wood will be 18 inches wide, 2 feet high, and 25 feet long. Stones are laid up over the pile and a space is left to light the fire. The fire is lighted at 5 p. m. and left to burn until 8 a. m. the next day. The stones, which quickly get hot, are regarded as most efficient in thawing. On a 4-foot thickness of pay this amount of fire will thaw, in the time specified, from 5 to 6 cubic yards of gravel. This is at the rate of 9.2 cubic yards thawed to the cord of wood, which is considerably less efficient than the method of thawing with steam. Time, delays, and awkwardness of the method, moreover, make wood-fire thawing the most expensive that can be adopted. The figures per foot for shaft sinking range from $3.16 to $7.50 in taking gravel from prospect shafts. The efficiency given in the above-cited case on Deadwood Creek appears to be very nearly the same as that given by Mr. E. D. Levat[a] for the operations in eastern Siberia.

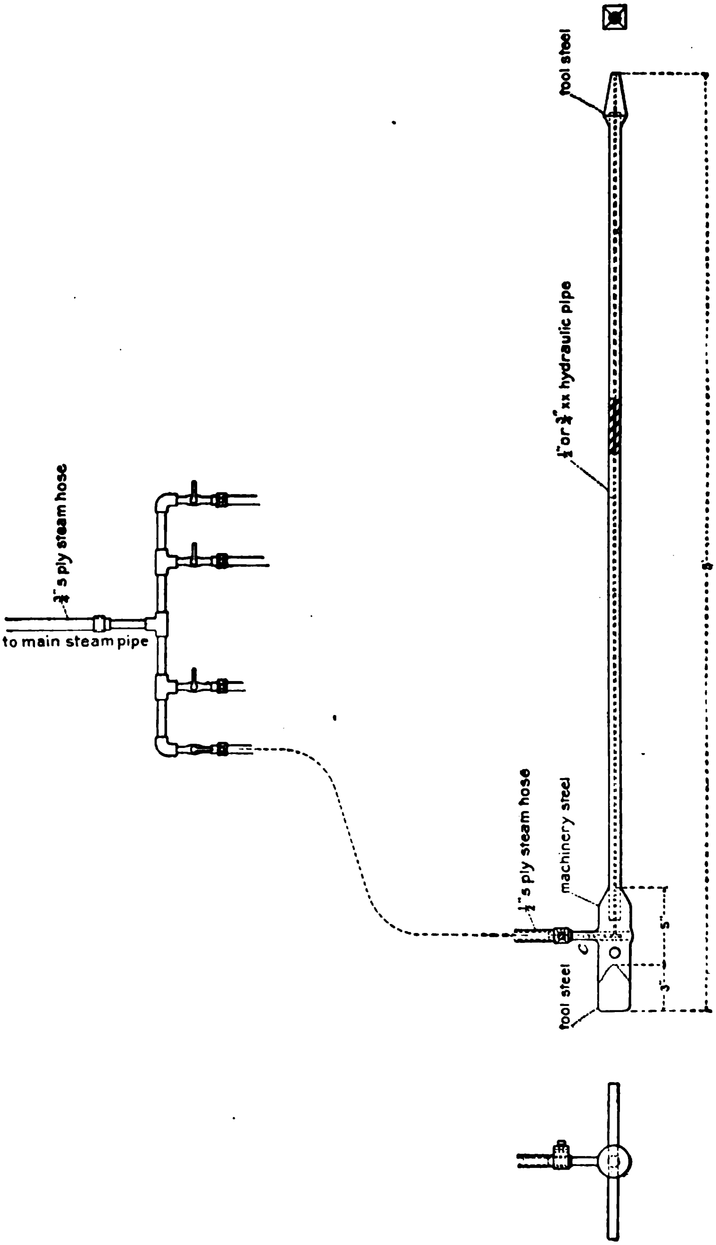

FIG. 17.—Details of steam point and crosshead.

The direct application of jets of dry steam to the gravel bank through the agency of driven pipes has been found to be the most efficient method in general practice for thawing frozen gravel. The amount of moisture contained in steam can be judged by the color of a jet of steam issuing from a small brass pet cock. If it is transparent or whitish near the orifice it contains less than 2 per cent of moisture; if pure white, the moisture is above 2 per cent.

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[a] *L'Or en Sibérie orientale.* The formula used is: Each breadth of 1 inch of wood applied at the *foot of the face* thaws an equal width of gravel.

Fig. 17 shows the details of the steam point and some of the fittings which its use entails.

In creek claims exceeding 15 feet in depth where solidly frozen ground occurs, as, for example, in the new Fairbanks district of Alaska, the method of drifting with the use of the steam point is as follows: A 20-horsepower boiler, capable of running 10 steam points, is put on the ground, and frequently one or two extra long points are provided for sinking holes. These long points, from 10 to 12 feet in length, are not so strongly made as the 5-foot points used in the drifting operations. In some cases pieces of one-half-inch hydraulic pipe are used. The point is set up on the ground and steam or hot water is turned on. The time for sinking a hole by this method to bed rock is from twenty-four to forty-eight hours. If large, flat stones are encountered in the gravel underlying the muck it is sometimes advisable to use strong, specially made points to prevent breaking. The average radius of a vertical shaft thus thawed by a single point is 2 feet, and the hole when cleaned out has a cylindrical or tube shape.

If timbering is required after the shaft is sunk it is generally found that the hole will thaw sufficiently to allow the shaft to be carried 4 by 5 feet or 4 by 6 feet inside the timbers. As a rule, however, winter shafts are not timbered. When bed rock is reached a tunnel or runway is run for a distance varying from 50 to 100 feet from the tunnel lengthwise of the claim. From this central drift, which is generally timbered, lateral drifts are run to a distance not exceeding 50 feet, or from this down to 10 feet, or the width of the pay streak. Two tunnels parallel with the central runway are carried at the ends of the cross drifts and connections made from them to the shaft. These are rarely timbered.

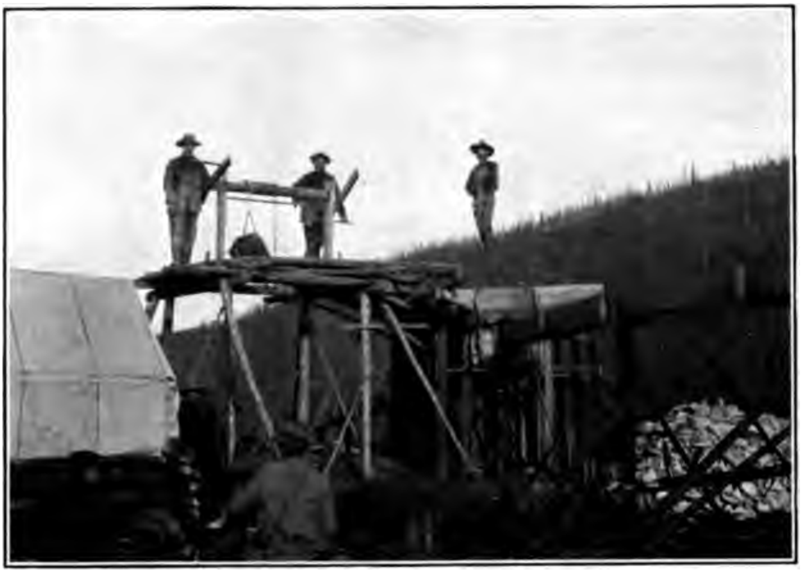
The frozen condition of the overburden allows a firm roof to stand without timbering while the gravel is being extracted. This is a great advantage and in part offsets the difficulties attending the breaking up of the frozen gravel. The ground is now ready for working. The drifts and main ways are run as low as possible in cases where the pay is thin, but a height of 3½ feet is the lowest that can be worked with economy.

During the night shift sufficient gravel is thawed to occupy the men in extracting it in the daytime. In handling, not only the picking and shoveling from the bank must be allowed for, but the wheeling to the shaft, the time of return to the face, and the hoisting of the gravel to the surface and conveying to the dump. The face exposed by the cross drift at the far end of the runway is worked back toward the shaft, and the gravel is trammed in wheelbarrows to the shaft and hoisted. Points 5 feet in length are used to thaw the gravel, and are generally used in "batteries" or "crossheads" of four (see fig. 17, p. 90). The points are supplied with steam from a crosshead of iron gas pipe, three-fourths inch in diameter, fitted with elbows and short pieces of one-half inch pipe leading to the steam hose, to which the points are attached. The valves are generally set in these short pieces of pipe. The points are driven in with a mallet by the point man working on the night shift. The points are left in the bank from ten to fourteen hours. The thawing is nearly always done on the night shift, the only men occupied being the point man below and the fireman at the boiler above. Each point thaws a block of gravel on an average 6 feet into the bank, 18 inches on each side of the point and 4 feet high. The crosshead is laid on the floor of the drift, at a distance of 10 feet away from the face, and the steam hose connecting each cock of the crosshead with the individual points must be long enough to reach the desired distance for placing the point in the bank. The cross pipe *c*, to which the hose clamp attaches the hose, should be carried through the steel head and brazed on the opposite side so that it will not break off by a chance blow from the mallet.

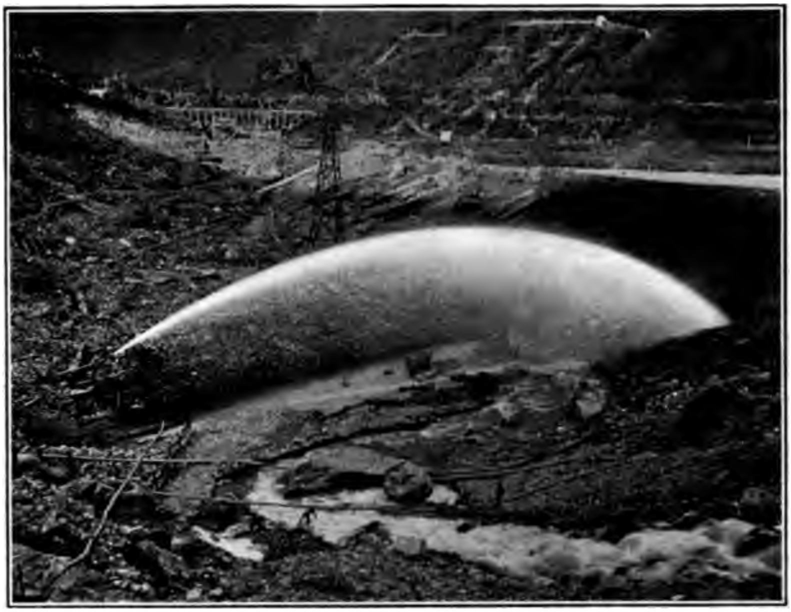
It is considered good practice as a rule to start the points with hot water turned through the hose. The points must be driven carefully and slowly, and for 10 points distributed along a face the average time needed is from one to three hours. The amount of steam required for each point has been found to vary in amount from 1 to 2 boiler horsepower. One and one-half may be taken as a safe average. The amount of gravel which a point will thaw appears to vary with the length of the point, and this is regulated somewhat by the character of the gravel. The 5-foot point has been found the most economical size for the small operator.

A typical illustration of the efficiency of the points in the Fairbanks district is the experience at No. 8, below Fairbanks Creek. Points of Dawson manufacture, 5 feet long, costing $15 each laid down, were used in crossheads of four. They were put in at distances of from 2 feet 6 inches to 3 feet apart. The points were started with hot water, and it took three hours to drive them in. A 12-horsepower boiler supplied the steam for 10 points, three-fourths of a cord of wood being burned on the night shift, when the thawing was done. In twelve hours the 10 points thawed a block of gravel 30 feet in length by 5 feet in height by 6 feet into the bank—an average of 3.3 cubic yards to the point.

The above is a little high for the average duty in the Fairbanks district, but is low for the Klondike creek drifting operations. A typical



*A.* DRIFT MINING ON FAIRBANKS CREEK.



*B.* HYDRAULICKING, SILVER BOW BASIN.

result on Hunker Creek, where 1¼ to 1½ horsepower was used for each 5½-foot point running ten hours, was 4½ cubic yards. Pl. III, *B* (p. 40), shows a winter drifting operation in the Klondike, and Pl. XV, *A* (p. 92), shows a drifting operation during summer on Fairbanks Creek. Pl. XIV, *B*, shows a battery of steam points thawing ground in the open in preparation for excavating by a steam shovel. Three cords of wood, at $13 per cord, were burned in twenty-four hours to generate steam for operating 15 of these points 10 feet long. These thawed ground 14 feet deep to bed rock. As 4 cords of wood were burned in addition on the steam shovel, and 20 men, at $6 a day, were employed in the twenty-four hours to get out less than 500 cubic yards a day, the operations were not economical. In connection with a dredging operation the thawing by means of steam conducted through gas pipe in 11-foot lengths was estimated to add 40 cents per yard to the cost of working the ground.

The only drifting and steam-thawing plant at Nome gave an efficiency for each point of only 1¼ cubic yards. The points were using in this case a little less than 1 horsepower each and were run only for eight hours. Moreover, the gravel was small and was mixed with much ancient beach sand, which in a frozen condition is extremely difficult to thaw with steam.

The method of thawing with hot water by means of a force pump set in the underground workings, which forces hot water through a small nozzle against the bank, has been tried successfully in the Klondike district. At a claim on Gold Run, where it was desired to extract a 3-foot pay streak of gravel capped by 27 feet of barren gravel at a depth of 50 feet below the surface, a small force pump of the ram pattern, with outside packed valves, was placed in the main runway near the shaft. It drew water from a 6-foot sump near at hand, to which the workings drained. The pump had 4-inch intake, 3-inch discharge choked to 2½-inch, and the water was pumped to the face by means of cotton hose and discharged through a 1-inch brass nozzle at 40 pounds pressure. Six thousand gallons of water were used over and over, and by discharging the exhaust from the pump into the suction the water was kept at a temperature of 150° F. In a shift of ten hours the pump, using 30 horsepower, thawed and broke down ready for the shovelers 175 cubic yards of gravel. This is vastly superior to the average Klondike duty of the 1½-horsepower steam point, even allowing 4 cubic yards to the point, as the 30 horsepower would supply only 20 points and the maximum duty would be 80 cubic yards.

Only under certain very favorable conditions, however, can the hot-water method be used. There must be no silt in the gravel, otherwise the water becomes thick and can not be settled in the sump. If the bed rock is in large flakes, and the pay sinks into it, the hydraulic method is inefficient in thawing the valuable ground. The method is worthy of the serious attention of the Fairbanks placer miners, however, as will be pointed out in the discussion of that district (p. 97).

## HOISTING.

For hoisting and conveying, say to a distance not exceeding 200 feet, the Dawson self-dumping carrier, with accompanying bucket, cables, etc., and steam hoist, must be used in order that the most economical work per man employed can be done. This device, which is shown in fig. 18, is without doubt the best system of handling gravel from a shaft of shallow depth now in use in the Northwest. It is strong and compact, and is simple in operation. Those parts which by their position receive the most strain and jar can be made sufficiently durable to withstand hard usage.

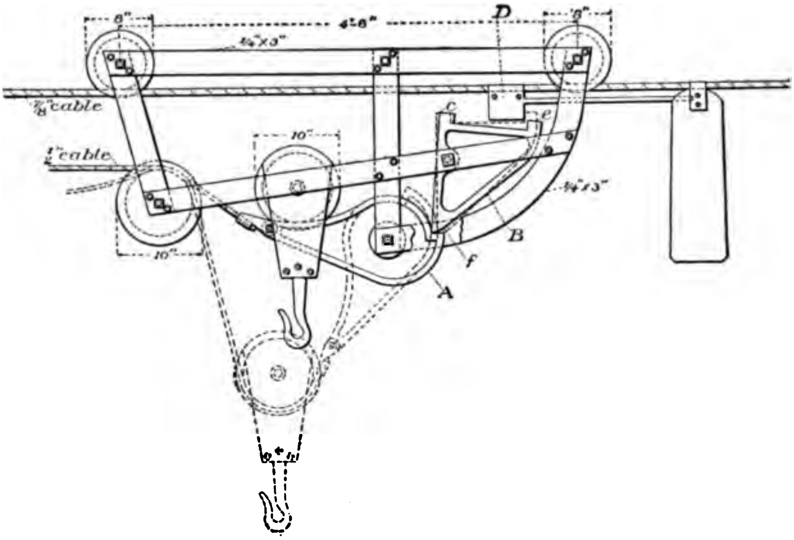


FIG. 18.—Dawson self-dumping carrier.

No springs are used in the construction, for the reason that at low temperature, say of 40° F. below zero, metals are very brittle, and the constant jar, a necessary accompaniment of the work of this apparatus, renders springs and any light metallic member unsafe.

As fig. 18 shows clearly the construction of the carrier, only a few words need be said of its operations.

It may be stated that three distinct operations are accomplished, two of which are performed by the carrier, the third being effected by an auxiliary rope used in dumping the bucket. The first consists in engaging the bucket as it arrives from the shaft and carrying it to the dump box, the second in returning the bucket to the head of the shaft and there dropping it to the bottom. As the carrier returns down the seven-eighths inch cable, the hook to which is attached the bucket occupies the position indicated by the solid line. The cam, A, lies horizontally and is held firmly in this position by the dog, B, the weight of the bucket pressing the notch of the cam against the point of the dog, *f*. When the carrier reaches the head of the shaft the end, *c*, of the dog strikes the block, D, and frees the point of the dog from the notch of the cam. The front of the cam now occupies the position indicated by the dotted line, and, as the bucket sinks into the shaft, is pressed against the block, since the carrier tends to move backward. Thus as the bucket sinks with the hook on the one-half inch cable the carrier is held firm.

As the bucket rises from the shaft or pit, the strain is toward the engine, and the lower end of the dog, *e*, still prevents the apparatus from moving.

When, however, the cam, A, by the upward movement of the bucket, reaches the horizontal position indicated by the solid lines, the point of the dog, *f*, jumps into the notch, lowering at the same time the point, *e*, which allows the carrier to move up the seven-eighths inch conveying cable, and also once more secures the cam in its horizontal position.

The block attached to the cable is held in a vertical position by a suspended log or block, which insures its engaging with the dog.

By this system of working, if the self-dumping rig handles 250 of the 30-pan buckets in ten hours or, roughly, 60 cubic yards, the plant will necessitate the employment of 12 men, namely, 2 firemen, 1 hoist man, 1 point man, and 6 men shoveling and wheeling, beside the foreman. The ordinary set-up of the self-dumper is shown in fig. 13. The boiler used for generating the steam for points at night is used for running the hoist in the daytime. In running from 15 to 20 points at night it will burn a cord of wood, and in hoisting in the daytime will burn from one-half to 1 cord more. The total daily twenty-four-hour expense can not be brought below $130 and will probably amount to $150. It is evident that with such a plant the tenor of the gravel must amount to at least $2.60 to the cubic yard to pay any profit at all, exclusive of washing in the spring, and should be 4 cents to the pan, or $5.20 to the yard, to pay a profit which compensates for operating in so remote and expensive a country. The cost of washing in the spring is in general from 50 cents to $1 per cubic yard.

The winter dumps are piled in a conical heap around the gin pole, to which the trolley cable of the self-dumper has been attached while the dumping is in progress. In laying out the space for the dump, the string of boxes in which the gravel is to be washed during the following spring is so laid that it will bisect the proposed cone, having its proper grade and length. When the spring opens the dump is easily shoveled into the boxes, from both sides, beginning at the lowest point and working upward. Boards are generally laid over the top of the boxes, and a portion of the gravel rests on these. The boards are taken off as the work of shoveling in progresses.

The expenses of winter work are much greater than those for summer, and the tenor of the gravel must in consequence be much higher. In the first place it is to be remembered that the gravel dumped in winter must be all rehandled in the spring. In thawing also the amount of steam used in winter is greater than in summer, and all pipes and hose conveying steam both above and under ground must be coated with asbestos or other nonconducting material. On the other hand, labor is generally cheaper in winter than in summer, and the possibility of striking thawed streaks of ground is much less.

### DRIFTING IN THE FAIRBANKS DISTRICT.

As has been stated, the only mining practiced at Fairbanks is creek mining. In the open season, although the bulk of the ground can be worked by drifting, a smaller proportion is workable by open cutting. By far the larger portion of the ground must be worked by drifting on account of the depth to bed rock and, in many cases, the thinness of the pay. The ground is in the main frozen, but thawed streaks occur. The gravels are of the angular and subangular character which marks all the gravels deposited in recent time in the northwest. The angularity is more pronounced than in the Birch Creek district or the Klondike.

One characteristic feature of the Fairbanks gravel has an important bearing on the method of thawing the frozen gravel. This is the fact that while the layer of so-called muck is comparatively thin, from 2 to 7 feet in the workings on the creeks themselves, exclusive of the slopes, the gravels, which are from 6 to 15 feet in thickness, do not carry pay throughout their section, but only in the lower part. The pay is sometimes as thin as 6 inches in the gravel, and rarely exceeds 3 feet. It is then frequently necessary to take only a portion of the gravel down, and to run the drifts as low as possible, i. e., not to exceed 3½ feet. To do this economically, and so that the top and barren gravel will not be continually caving and falling during the drifting, the ground must either be timbered or the method of steam thawing must be abandoned. Another reason why the method of steam thawing has a limited application in the Fairbanks district is the fact that the gravel is very argillaceous in its pay portion, and the thawing and drying with the steam result in a baking of the ground. Thus the gravel which was frozen becomes in its dry state cemented, and the difficulty of getting it out is not avoided but only lessened.

Thawing by means of hot water driven through a force pump and conducted to the bank by means of cotton hose and piped against the bank by means of a small fireman's nozzle is the most successful method which has been so far tried to thaw ground under the above-mentioned conditions. As regards the heating of the water, the system adopted at Fairbanks is to conduct the exhaust steam of the pump back into the suction of the pump as it draws water from a sump alongside of it. This has been found more effective than simply turning the exhaust pipe directly into the sump itself. The water used is sometimes supplied in sufficient quantity from the mere seepage from the ice mixed with the thawed gravel. But sometimes it will have to be introduced in the pit in small quantity. On the other hand, especially where thawed streaks occur naturally in the gravel, the water must be pumped out. This is done by the same pump used for the hydraulicking, without changing its position.

This method of thawing by hot water piped against the bank has two advantages. The strongest appears to be the fact that gravel thawed by this method can, as it were, be selected in the face, and no higher gravel than may be profitable need be taken down. The portion of the gravel above the part taken down remains solidly frozen, and consequently the roof of the drift or wide stope does not cave any worse than does the roof of solid muck in drifting operations where all the gravel in its complete vertical section is extracted. Another advantage is that the gravel is moved from its original position farther than by steam, and if it contains clay the clay has less opportunity to bake and cake than with steam thawing, and the amount of hand work in picking down the gravel is lessened.

A serious objection to thawing by hot water may justifiably be raised in cases where the bed rock consists of large slabs of schist, into which the gold sinks from 1 to 4 feet and even more. In such a case the water can not be piped to reach all the frozen gold-bearing material. Although I am not aware that a combination of the hot-water and steam-thawing methods has been tried in such a case as the above, it seems not impossible that such a combination would be the most economical. On the other hand, where the bed rock consists of finely comminuted schist or softened, thoroughly rotten rock, even should the gold be found in it to the depth of 2 feet, there can be no valid objection to the hot-water hydraulicking method. In the Nome district of Seward Peninsula, where the hot-water method was tried, it was found that the water finally became too thick with sediment to be used in the pump, and the system was abandoned.

A minor recommendation for the use of the hot-water method is the fact that all the unpleasantness of the steam method, which fills the workings with steam, creates dampness, etc., is obviated, and the walls and roof of the drifts are dry, while ditches cut in the bed rock on grade to the pump sump can be so cut as to prevent the floors from being wet. The obscurity of the air in the workings by steam, especially in the early part of the day shift, is frequently a source of danger, as the roof can not be watched so closely as it could were the air clear, and consequently slabs of the roof that are ready to fall can not be easily seen, so that the liability that men will be crushed is greater.

I should especially recommend the method of hot-water thawing to the consideration of the miners of Cleary and Pedro creeks, while on Fairbanks Creek, as hitherto developed, the method of thawing with steam points seems more applicable.

In the Fairbanks district shafts sunk for work in the summer time are in nearly all cases timbered, either wholly or partially. In very deep shafts, as, for example, on the left bank of Cleary Creek near the junction with Chatham, where 40 feet or more of solid muck is encountered, timbering does not appear to be always necessary throughout the whole of the shaft. The muck forms a wall which resembles in its consistency a wall of solid ice, and in working throughout a season of four months, with the shaft in constant use, there will be little caving of the walls. All shafts that extend 20 feet or less, however, especially where gravel composes half or more of the section—shafts intended for working out a block of ground, conducting steam connections, pump connections, and connections for hoisting—should be securely timbered.

In view of the local conditions, the following method of timbering appears the most satisfactory. After the hole is sunk to the required depth by thawing, a square set with 6-foot centers or should the shaft be 4 by 6 feet, of this dimension is put in at the bottom. Three-inch pole lagging, with cross sets of 6-inch timbers inside at 6-foot intervals, is carried up to the collar. Inside the sets 6- or 12-foot poles, 3-inch, are nailed in the four corners to the cross sets. Moss is laid up outside the pole lagging, as dry as possible, and packed against the gravel or muck walls. In cases where it is impossible to sink without timbering as the shaft is sunk the timbering is carried down, and the lagging is driven as each set progresses, the square set being put in at the bottom on the completion of the shaft. Shafts timbered in this manner have been found to stand for periods of from one to two years, as long as the work in the Fairbanks district has been in progress, and there is no reason why they should not stand much longer. Timbering of this sort can be done for the most part with the ordinary cord wood delivered for fuel, and this is much the cheapest method available.

The cribbing method is sometimes adopted, accompanied by the same packing, with moss outside the cribbing. It is, however, more expensive, and in a shaft of considerable depth is much more likely to get out of plumb in this frost-ridden country.

In the Fairbanks district it has not yet been found necessary to timber the underground workings to any great extent. The main runway, if it is to be used for the whole season or for a period exceeding a month, should be timbered for safety. Six-foot sets of posts and caps only, with lagging over the roof, are generally sufficient. Pole lagging is cheaper and more advisable than split lagging. The caving system, working always toward the shaft, allows the faces to be carried forward without timbering of the stopes. The elaborate timbering methods necessary in the Klondike benches and in many parts of California in drifting operations are happily not needed in this portion of Alaska, and are not likely to be required in any extension of the Tanana area. No case has been seen in the operations, as thus far developed, where false sets were carried in taking out the gravel. When the operations at Fairbanks become more extensive, however, and drifting is carried on more systematically with wide faces, it is very likely that the workings will require timbering. At present it is not possible to give the cost of this work, as the figures were not obtainable.

## HYDRAULIC MINING.

### GENERAL REMARKS.

The hydraulic method of working gold gravel deposits is the most economical method from the standpoints of power, capacity, and labor.

It is, on the other hand, exceedingly wasteful of water, and the topographic conditions under which it may be successfully and profitably conducted are not of common occurrence, even in mountainous districts.[a]

The hydraulic method of mining was invented by Edward E. Matteson, a native of Sterling, Conn., and applied by him to the auriferous gravels in California in 1852.[b] One or more jets of water under pressure are thrown from a pipe or pipes with great velocity against the face of a gravel bank, and the gravel is loosened by the stream so that it falls or "caves." The gravel is struck with sufficient force to be disintegrated and broken up so that it can be carried by the current into the sluice if the bed rock has a sufficient grade. For assistance in moving and washing the gravel after it leaves the bank, additional water is frequently used under slight head, and is known as "bank head" or "by-wash" water. Various devices for more easily disintegrating the bank and for disposal of the large bowlders and of tailings where natural grade is lacking will be later discussed.

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[a] The amount of water used in hydraulicking is shown by the following illustration: Three hundred million gallons of water a day are used at present for the supply of Greater New York City, or 26,666 miner's inches, of twenty-four-hour service. Taking the duty of an inch at 2 cubic yards, and a tenor of 15 cents in gold per cubic yard, the above amount would supply only 13 hydraulic mines, each using 2,000 miner's inches under head. The total amount of gravel washed would be 52,000 cubic yards per day, and the product $7,800; or (with a season of one hundred and twenty days) annually, 6,210,000 cubic yards, and a product of $936,000, an amount not startlingly large, considering the magnitude of the operations involved.

It is to be expected that persons with a limited knowledge of hydraulic mining will object to the low capacity and short season assumed in the above, but operators of long experience will realize that the conditions and results are assumed, if anything, more favorable than is found in average practice.

[b] Whitney, J. D., The auriferous gravels of the Sierra Nevada of California: Mem. Mus. Comp. Zool. Harvard Coll., vol. 6, 1880.

In California the application of hydraulic mining spread with great rapidity, especially within the drainage basins of San Joaquin and Sacramento rivers. This method was adopted also in other portions of the western United States and in other parts of the world, particularly in Australia and New Zealand. In the part of California referred to, extensive operations ceased, not because the gravel deposits were exhausted, but because of economic considerations other than those of mining. The passage of an act of Congress in March, 1893, entitled "An act to create the California Débris Commission and regulate hydraulic mining in the State of California," sounded the death knell of the huge mining operations which had changed the topography and blanketed many square miles of arable lands in the San Joaquin and Sacramento valleys with gravel tailings.

The comments of J. D. Whitney,[a] written in 1880, concerning the application of hydraulic mining to California conditions, are as fcllows:

> It is owing to a happy combination of favorable circumstances that the system of hydraulic mining has been so successful on the slopes of the Sierra Nevada. That the peculiar set of conditions which makes hydraulic mining possible is not often met with is sufficiently proved by the fact that this system, which seems so admirably adapted to the needs of the Californian gravel miners, has hardly been at all successful in any other region. It has been tried again and again in the southern United States with almost unvarying loss; and even in Australia, where the mode of occurrence of the gold is in many respects so similar to what it is in California, there are few districts where the hydraulic method can be applied.
>
> The first great need of the hydraulic miner is an abundance of water and with a considerable "head," so that the stream may issue with sufficient velocity from the pipes.
>
> An abundance of water can not be secured without extensive engineering operations and the expenditure of a large amount of money. Extensive reservoirs must be constructed by building dams across the outlets of the mountain valleys, so as to impound the water coming from the melting of the winter's snow on the high Sierra, and the necessary canals—or ditches, as they are universally called by the miners—must be excavated to carry the water to the points where it is needed for use. The long, rapid, and rather uniform slope of the Sierra, in the mining districts, makes it possible almost everywhere to carry the ditches with such a grade and in such a position as to allow the water to be taken from them at a sufficient elevation to give the necessary head at the point of working. The great elevation of the important gravel masses and the deep canyons into which the whole mining region is cut up, afford, in almost every locality, the necessary facilities for arranging the sluices and disposing of the tailings.

A need equally as great, not sufficiently emphasized by Whitney, is adequate grade or slope of the bed rock for the moving of material. Experience has shown that the lightest grade of stream bed upon which hydraulic operations may safely be undertaken is 220 feet to the mile.

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[a] Loc. cit., p. 62.

The above statements regarding hydraulic mining are as worthy of consideration to-day as when they were written. Bench deposits, as already defined, are those above all others to which the hydraulic method is most likely to be successfully applied, because adequate grade for the disposal of tailings can be secured. Deposits in the beds of present streams, on the other hand, are exploitable by hydraulicking to a far less extent. In Alaska extensive bench deposits do not occur, even in the alpine districts of the south coast, and, moreover, where auriferous benches are found they are not backed by high mountains from which the water under pressure can be obtained. The bench deposits of Alaska are not characterized by great thickness as are those of California. A depth of 10 feet of auriferous gravel with 20 feet of overlying barren silt, or a total of 30 feet, is a not uncommon condition in Alaska, while in California gravel banks of 75 to 500 feet in thickness occur and have been worked by hydraulicking. The hydraulic operation shown in Pl. XV, *B* (p. 92), is on a lake bed, locally formed in a portion of Silver Bow basin, Juneau district.

The hydraulic principle implies the breaking down of all the alluvial material by water under pressure, the cheapest power. It is evident that unless the gold is distributed throughout the vertical section of the alluvial deposit, as well as on and in the bed rock, the greatest amount of material is hydraulicked off at a loss. In general the pay dirt in Alaska composes less than 15 per cent of the bank, measured vertically. In California gravels, on the other hand, much more frequently a small amount of gold occurs throughout the whole bank.

In Alaska attempts have been made to work very shallow ground, in some cases of 5 feet section, by hydraulic methods involving the construction of ditches and installation of machinery at a cost of $100,000 and over. Unless the gravels worked were much richer than previous workings in the vicinity, it was impossible to see how the enterprise could by any means be profitable. Simple mechanical plants, requiring a comparatively small investment, could have handled the small body of gravel fully as fast and in the long run more cheaply. It is undoubtedly true that many men are deterred from installing really economical plants of a mechanical nature by the desire to apply the spectacular methods of the hydraulic miner, without regard to the existing conditions.

A word should be said about the application of power to produce pressure for hydraulicking.[a] Several plants where this principle is applied have been visited in Alaska and the neighboring British territory. Nine plants were seen and it was not clear that any plant pumping water for hydraulic mining was making expenses. Some Nome operators are of the opinion that if crude oil can be obtained for $2 a barrel the gas engine can be profitably used to hydraulic certain rich patches of the coastal plain gravels, using 50 to 60 pounds pressure at the nozzle. Only under the most exceptional conditions would such an application of the hydraulic method be profitable. It may be truly said that 95 per cent of all attempts to pump water in hydraulic gravel mining, in whatever portion of the world they have been tried, have been failures. The miner who contemplates such an operation in a remote region where coal costs $30 a ton or wood $15 a cord had better first exhaust the possibilities of all other methods of moving gravel which circumstances or ingenuity can suggest.

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[a] These statements do not refer to the hot-water method of hydraulicking frozen gravel.

It must not be assumed that the tenor of the gravel in Alaska is greatly above that of the auriferous gravels formerly worked so extensively in California, especially where it is necessary, as in hydraulicking, to distribute the value contained in the thin-bottom pay streak throughout the whole of the vertical section of gravel which must be moved.[a] There is a widespread opinion, amounting to a dictum, that the hydraulic method should be installed in preference to others wherever there is the slightest chance for its success. That this chance is small in the hitherto developed portions of Alaska the present account will make clear. Capital invested by the miner in expensive water conduits in Alaska is not recoverable except from the profit on his operations. The ditch builder can not enter the domain of commerce and sell his water or the power generated thereby, as do many of the water companies in populous agricultural and industrial communities. In rare instances only can he even sell it to miners. On the exhaustion of his own auriferous ground, his ditch and plant becomes practically valueless.

While hydraulicking river and creek beds in Alaska with the necessary accompaniment of some form of lifting operation for the tailings is not to be entirely decried, it should be undertaken with extreme caution. The ever-present danger of floods and the expense of wing dams must be considered. The tailings will be raised by the water lift primarily, and the method of using this in gravel mining will be fully described. The operator should bear in mind, however, that the introduction of the water lift decreases by over 60 per cent the duty of the unmodified hydraulic method as above described; also that the use of the ingenious device entails a new set of difficulties with which his previous experience in hydraulic mining will hardly have fitted him to deal. An unfortunate instance was seen in one of the remote districts of Alaska, where a hydraulic elevator of 5 tons weight had been imported at a cost of nearly $2,000 and was found useless because insufficient water to operate it was obtainable, although a ditch and flume line had been constructed at a cost of $15,000. The elevator lay on the bank, and a makeshift had been improvised from a piece of sheet-steel pipe and a fireman's brass nozzle.

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[a] For example, Whitney states (loc. cit., p. 118) that the area hydraulicked off at Todds Valley, Placer County, Cal., was 1 mile by one-fourth mile in area, and the gravel from 35 to 75 feet in thickness, a total of nearly 9,000,000 yards, with an estimated yield of $4,000,000, or about 44 cents per cubic yard.

At the Excelsior claim, near Placerville, Eldorado County, Cal., 20 acres of ground were washed off with an average thickness of from 40 to 60 feet of pay gravel. The yield was estimated at $5,000,000, about half of which was taken out by drifting and the other half by hydraulicking. Mr. Alderson, the principal proprietor, estimated that the yield of the Excelsior gravel was $1 per cubic *yard. A single placer* claim of 20 acres, containing so large an amount of gold as the Excelsior claim, *has yet to be* discovered in Alaska.

That hydraulic operations are and have been successfully prosecuted in Alaska is not to be denied. The cautionary remarks which have been made will serve as a reminder, however, that the moving of large bodies of gravel by water under natural head is not always as cheaply accomplished as a casual inspection of such operations would induce one to believe. In any event, a hydraulic installation, under conditions where the cost of leading water runs into the thousands of dollars per mile, should not be undertaken until due consideration as to the possibility of using other methods has been given.

## DITCHES, FLUMES, AND RESERVOIRS.

In table 8 an attempt has been made to show in concise form the principal facts regarding the construction of water conduits in Alaska and the North. So great was the extent of territory covered that opportunity for measurements and calculations was only occasional. The statements of operators were generally accepted; they were rejected only when they were palpably misleading. When not otherwise indicated, the data shown were supplied by the operators. When the data were based on subsequent estimates, the figures are starred.

It can not be too strongly stated that the tables in this report are based on statements of operators and are not the result of detailed investigation of properties. The collection of data for this report necessitated the covering of a vast extent of territory in four months' time. The data are as complete and as nearly accurate as the circumstances allowed. Estimates based on the cost of water conduits as given below, and the amount of water available, should take into consideration the degree of authority which attached to the compilation.

TABLE 8.—*Water conduits in Alaska and the North.*

| Locality. | Ditches. | | | | | | | | | | Notes. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Length earth work. | Length rock work. | Grade per mile. | Width. | | Depth. | Average amount of water used, miner's inches (1.5 cu. ft. per min.) | Greatest head. | Cost of construction per mile. | Annual cost of maintenance per mile. | |
| | | | | Top. | Bottom. | | | | | | |
| | | | | *Feet.* | *Feet.* | *Feet.* | | *Feet.* | | | |
| South Coast Alaska: | | | | | | | | | | | |
| Douglas Island | | | | | | | | | | | |
| Gold Creek | | | | | | | | | | | |
| Do | | | | | | | | | | | |
| Silver Bow basin | | | | | | | | | | | |
| Windfall Creek *a* | | | | | | | | | | | |
| Lemon Creek | | | | | | | | | | | |
| Sunrise district: | | | | | | | | | | | |
| Miller Creek *a* | 2 miles | | 18 feet | | | | 1,500 | 110 | | | |
| Crow Creek *a* | 1 mile | | 26 feet | 5 | 3 | 2 | | | | | |
| Atlin, B. C.: | | | | | | | | | | | |
| Pine Creek | 4½ miles | | 8 feet | 12 | 6 | 4 | 1,500 | 135 | $3,520 | | |
| Do | 5¼ miles | | 8 feet | 9 | 4 | 3 | 1,000 | 162 | 3,000 | | |
| Do | 3 miles | | 8 feet | 7 | 4 | 3 | 700 | 350 | | | |
| Birch Creek | | | | | | | | | | | |
| Spruce Creek | 2,000 feet | | 5 feet | 9 | 4 | 5 | 900 | 200 | | | |
| Do | 1¼ miles | | | 7 | 4 | 2½ | 1,200 | 200 | 5,200 | | |
| McKee Creek | | | | | | | | | | | |
| Do | | | | | | | | | | | |
| Do | | | | | | | | | | | |
| Do | 3,300 feet | | 26 feet | 9 | 3 | 2½ | 1,000 | 325 | 5,220 | | |
| Porcupine district: | | | | | | | | | | | |
| Porcupine Creek *a* | | | | | | | | | | | |

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Dawson district, Y. T.: | | | | | | | | | | | |
| Miller Creek[a] | 9¼ miles | | 8 feet | 8½ | 3¼ | 3½ | 1,200 | | | | |
| Do | 8.6 miles | | 8 feet | 5¼ | 2¼ | 1¾ | 600 | | | | |
| Bonanza Creek | | | | | | | | | | | |
| Do | 9,400 feet | | 13.7 feet | 5 | 3 | 1½ | 100 | | | $350 | Cost, ditch, flume, and siphon, $12,000. |
| Do | 2 miles | | 7 feet | 5 | 2 | 3 | 150 | 150 | | | |
| Do | 8 miles | | 8 feet | 6 | 3 | 2 | | | 1,500 | | Ditch under construction. |
| Eldorado Creek | 6 miles | | 10 feet | 8 | 4 | 3 | 120 | 120 | 3,000 | | |
| Hunker Creek | 4 miles | | | 3 | 2 | 2 | 150 | 20 | 2,500 | 150 | |
| Do | 5¼ miles | | | 3 | 2 | 1¼ | 100 | 40 | 2,000 | | |
| Eagle district: | | | | | | | | | | | |
| American Creek | | | | | | | | | | | |
| Fortymile district: | | | | | | | | | | | |
| Wade Creek[a] | 1 mile | | 8 to 9 inches | | | | 120 | | | | |
| Do | 400 feet | | 2 to 13 inches | | | | 200 | | | | |
| Birch Creek district: | | | | | | | | | | | |
| Deadwood Creek | 750 feet | | | | | | 120 | | | | |
| Do | 3,000 feet | | 6 to 8 inches | | | | 120 | 65 | 800 | | |
| Do | | | | | | | | | | | |
| Mastodon Creek | | | | | | | | | | | |
| Fairbanks district: | | | | | | | | | | | |
| Fairbanks Creek | 900 feet | | | 2 | 2 | 1 | 100 | | 5,800 | | |
| Do | 3,000 feet | | 4 to 5 inches | 5 | 4 | 1¼ | 150 | | 6,330 | 500 | |
| Cleary Creek | 500 feet | | 3 feet | 3 | 2 | 1¼ | 75 | | 2,500 | | |
| Do | 1½ miles | | 3 feet | 3 | 2 | 1¼ | 50 | | 3,200 | | |
| Do | 1 mile | | 3 feet | 3 | 2 | 1¼ | 50 | | 2,000 | | |
| Rampart district: | | | | | | | | | | | |
| Hoosier Creek[a] | 3,500 feet | | 3 to 4 inches | 8 | 6 | 1¼ | 350 | 75 | [b]10,500 | Expensive. | |
| Pioneer Creek[a] | 3¾ miles | | 3 to 4 inches | 5 | 4 | 1¼ | 300 | | [b]3,750 | | |
| Nome district: | | | | | | | | | | | |
| Nome River[c] | 17 miles | | 3.37 feet | 11 | 8 | 3 | 1,000 | 360 | 2,300 | 300 | $300,000 total cost, including annual maintenance for 4 years. |
| Do | 26 miles | 5 miles | 4.5 feet | 14 | 10 | 3 | [d]2,000 | | [e]3,000 | | |
| Do | 5 miles | 1,800 feet | 6.5 feet | 11 | 8 | 3 | 2,000 | | [e]2,300 | | |

[a] Not visited. [b] Ditch and flume. [c] See special description of the Miocene ditch (pp. 123–126). [d] On Aug. 23, 1904, measured 1,752 miner's inches. [e] Ditch.

TABLE 8.—*Water conduits in Alaska and the North*—Continued.

| Locality. | Ditches. | | | | | | | | | | Notes. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Length earth work. | Length rock work. | Grade per mile. | Width. | | Depth. | Average amount of water used, miner's inches (1.5 cu. ft. per min.) | Greatest head. | Cost of construction per mile. | Annual cost of maintenance per mile. | |
| | | | | Top. | Bottom. | | | | | | |
| Nome district—Cont'd. | | | | *Feet.* | *Feet.* | *Feet.* | | *Feet.* | | | |
| Basin Creek | 2¼ miles | | 10 feet | 7½ | 4 | 2 | 250 | 275 | $1,500 | | |
| Glacier Creek | 5 miles | | 5 feet | 4 | 3 | 1 | 115 | 25 | | | |
| Dexter Creek | | | | 5 | 3 | 1 | 120 | | | | |
| Cripple River | 4 miles | | 4.5 feet | 8 | 6 | 1 | 775 | | 2,500 | | |
| Council district: | | | | | | | | | | | |
| Ophir Creek | 4 miles | 1 mile | 4 feet | 9 | 6 | 2 | 1,000 | 192 | [a]4,000 | $330 | Cost, $8,000; includes 26-inch pipe line, 8,500 feet; total cost, $200,000. |
| Do | 13 miles | 5 miles | 3.3 feet | 16 | 10 | 3 | [b]3,300 | 200 | [c]6,000 | | |
| Do | 6 miles | | 3.3 feet | 8 | 5 | 2 | 600 | | 3,000 | | |
| Do | 1½ miles | | 5.3 feet | 9 | 6 | 2 | 150 | | 5,330 | | Includes rock cuts. |
| Do | 2½ miles | 400 feet | 3.3 feet | 9 | 6 | 3 | 1,500 | 100 | | | Total cost ditch, flume, and trestle, $30,000. |
| Do | 8 miles | | | | | | | | | | |
| Gold Bottom | 1 mile | | | 3 | 2 | 1½ | 100 | 60 | [d]1,000 | | |
| Solomon district: | | | | | | | | | | | |
| Solomon River | 9 miles | | 5 feet | 12 | 10 | 3 | | 125 | 8,000 | | Mile cost includes allowance three-fourths mile rock cut; 7½ miles more under construction. |
| Do | 6½ miles | | 4 feet | 16 | 12 | 5 | 1,200 | 120 | 15,000 | | |
| Port Clarence district: | | | | | | | | | | | |
| Sunset Creek [e] | 27 miles | | 5 to 8 inches | 15 | 12 | 1½ | 2,000 | | | | Ditch under construction. |
| Klondike River | 41,500 feet | 1,000 feet | 10 feet | 7 | 4 | 2½ | 500 | 300 | 4,330 | | Do. |

[a] Ditch; rock cut, $11,500.
[b] On September 11, 1904, did not exceed 1,600 miner's inches.
[c] Ditch; rock work, $8,000.
[d] Data estimated.
[e] Not visited.

TABLE 8.—*Water conduits in Alaska and the North*—Continued.

| Locality. | Flumes. | | | | | | | | | | | | | Notes. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Length. | Grade in feet per mile. | Width. | Depth. | Size of boards. | Size of posts and sills. | Size of stringers. | Average amount of water used, miner's inches (1.5 cu. ft. per min.). | Greatest head. | Cost of construction per mile. | Annual cost maintenance per mile. | Cost of lumber per M. | Average open season. | |
| South Coast Alaska: | | | Ft. in. | Ft. in. | Inches. | Inches. | | | Feet. | | | | | |
| Douglas Island | | | | | | | | | | | | | | |
| Gold Creek | 4,600 feet | 34.30 | 4 8 | 4 8 | 2 by 12 | 4 by 6, 4 by 4 | | | 200 | $11,616 | | $13.00 | May 15-Nov. 1 | Flume not in use. |
| Do | 1,000 feet | 36.90 | 4 0 | 2 0 | 1 by 12 | 2 by 4 | | 200 | 240 | | | 15.00 | 12 months | Water generates mill power. |
| Silver Bow basin | 3,000 feet | 27.50 | 4 0 | 3 0 | 2 by 12 | 6 by 6, 6 by 4 | | | 150 | | | 15.00 | May 15-Nov. 1 | Flume not in use. |
| Windfall Creek [a] | 5,000 feet | 14.00 | | | | | | 750 | | | | | Apr. 15-Oct. 20 | |
| Lemon Creek [a] | 2 miles | 26.00 | 5 0 | 3 0 | | | | | | | | 7.50 | May 1-Nov. 1 | |
| Sunrise district: | | | | | | | | | | | | | | |
| Miller Creek [a] | | | | | | | | | | | | | May 15-Oct. 15 | |
| Crow Creek [a] | | | | | | | | | | | | | do | |
| Atlin, B. C.: | | | | | | | | | | | | | | |
| Pine Creek | | | | | | | | | | | | | do | |
| Do | | | | | | | | | | | | | do | |
| Do | | | | | | | | | | | | | do | |
| Birch Creek | 1,960 feet | 10.00 | 3 6 | 1 7 | 1 by 8 | 2 by 4 | | 600 | 150 | | | 40.00 | do | |
| Spruce Creek | 2,500 feet | 5.00 | 4 0 | 2 8 | | | | 900 | 200 | | | 40.00 | do | |
| Do | Short sections. | | 4 0 | 3 0 | 1¼ by 8 | 4 by 6 | 4 by 8 inches by 12 feet. | 1,200 | 200 | | | 45.00 | do | |
| McKee Creek | 3,000 feet | 42.20 | 2 6 | 2 6 | [b]1¼, 1 | 3 by 4, 4 by 4 | 4 by 4 inches by 12 feet. | [c]2,000 | 170 | | | 45.00 | do | |
| Do | 3,000 feet | 36.90 | 3 4 | 2 8 | | | 4 by 4 inches by 12 feet. | 2,000 | 170 | 9,820 | | 45.00 | do | |

[a] Not visited.

[b] Bottom boards and side boards.

[c] May 20 to September 1, 2,000 miner's inches; September only, 500 miner's inches.

TABLE 8.—*Water conduits in Alaska and the North*—Continued.

| Locality. | Flumes. | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Length. | Grade in feet per mile. | Width. | Depth. | Size of boards. | Size of posts and sills. | Size of stringers. | Average amount of water used, miner's inches (1.5 cu. ft. per min.). | Greatest head. | Cost of construction per mile. | Annual cost maintenance per mile. | Cost of lumber per M. | Average open season. | Notes. |
| Atlin, B. C.—Cont'd. | | | *Ft. in.* | *Ft. in.* | *Inches.* | *Inches.* | | | *Feet.* | | | | | |
| McKee Creek | 3,500 feet | 33.20 | 2 8 | 2 8 | | | 4 by 4 inches by 12 feet. | 2,000 | 170 | $9,028 | | $45.00 | May 15–Oct. 15 | |
| Do | | | | | | | | | | | | | do | |
| Porcupine district: | | | | | | | | | | | | | | |
| Porcupine Creek[a] | 1,100 feet | 26.40 | | | | | | 3,330 | | | | 20.00 | May 1–Oct 1 | |
| (?) Creek[a] | 200 feet | | | | | | | 1,000 | 400 | | | 50.00 | do | |
| Dawson district, Y. T.: | | | | | | | | | | | | | | |
| Miller Creek[a] | 3,000 feet | 16.00 | 5 0 | 3 0 | [b]1¼, 1 | 4 by 4, 4 by 6 | | 1,200 | 210 | | | 60.00 | May 25–Sept. 25. | 4,640 feet inverted siphon, 24-inch steel pipe. |
| Do | | | | | | | | | | | | | May 1–Oct. 1 | |
| Bonanza Creek | 1,700 feet | 30.00 | 2 6 | 1 8 | 1 by 10 | 2 by 4 | | 200 | 65 | 4,750 | | | do | |
| Do | 2,600 feet | | 1 8 | 1 6 | 1 by 12 | 2 by 4 | | 100 | | | $150 | 60.00 | do | 2,200 feet inverted siphon, 10-inch pipe, at $1.55. |
| Do | Short sections. | 9.10 | 3 8 | 2 0 | 1 by 12 | 3 by 4 | | 150 | 150 | | | | do | 1,500 feet inverted siphon, 15-inch steel pipe. |
| Do | do | | 4 2 | 2 5 | 1 by 10 | 4 by 4 | | | | 11,880 | | 60.00 | do | Several inverted siphons, 20–26 inch pipe, 16–12 gauge, cost $2.50 per foot laid. |
| Eldorado Creek | 3,000 feet | 10.00 | 2 0 | 2 0 | 1 by 12 | 2 by 4 | ([c]) | 120 | 120 | 3,000 | | | do | |
| Hunker Creek | | | | | | | | | | | | | do | |
| Do | 3,960 feet | | 1 8 | 10 0 | 1 by 10 | 2 by 2 | | 125 | 50 | 3,000 | | 45.00 | do | |

| | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Eagle district:** | | | | | | | | | | | | | | | | |
| American Creek. | 7,300 feet | 11.50 | 2 | 8 | 2 | 0 | 1 by 8 | 2 by 4, 4 by 4 | | 250 | 150 | 10,800 | | 75.00 | June 1-Oct. 1 | Trestles, 97 feet high, 300 feet long; 22 feet high, 400 feet long. |
| **Fortymile district:** | | | | | | | | | | | | | | | | |
| Wade Creek[a] | | | | | | | | | | | | | | | May 10-Oct. 1 | |
| Do | 1 mile | | | | | | | | | | | | | | do | |
| **Birch Creek district:** | | | | | | | | | | | | | | | | |
| Deadwood Creek | | | | | | | | | | | | | | | June 15-Sept. 15. | |
| Do | | | | | | | | | | | | | | | do | |
| Do | 1,200 feet | 17.50 | 3 | 0 | 1 | 0 | 2 by 8 | 4 by 4 | | 120 | 20 | 1,320 | | | do | Built of flatted timbers hewn. |
| Mastodon Creek. | 2,000 feet | 2.00 | 2 | 6 | 1 | 4 | 1 by 8 | 2 by 4 | | 60 | | [d]5,000 | | 125.00 | June 15-Oct. 1 | Lumber whipsawed, at 15 cents per foot. |
| **Fairbanks district:** | | | | | | | | | | | | | | | | |
| Fairbanks Creek | | | | | | | | | | | | | | | May 1-Oct. 1 | |
| Do | | | | | | | | | | | | | | | do | |
| Cleary Creek | | | | | | | | | | | | | | | do | |
| Do | | | | | | | | | | | | | | | do | |
| Do | | | | | | | | | | | | | | | do | |
| **Rampart district:** | | | | | | | | | | | | | | | | |
| Hoosier Creek[a] | 800 feet | 3.33 | 6 | 0 | 1 | 6 | | | | 350 | 75 | [e]10,500 | | 120.00 | June 15-Oct. 1 | Oregon fir used; ditch in bad ground. |
| Pioneer Creek[a] | 2,000 feet | 3.33 | | | | | | | | | | [e]3,750 | | | do | |
| **Nome district:** | | | | | | | | | | | | | | | | |
| Nome River[f] | | | | | | | | | | | | | | | June 25-Oct. 10. | |
| Do | 1,110 feet | 9.00 | 8 | 0 | 2 | 9 | 1¼ by 12 | 4 by 6 p, 2 by 8 s | | 2,000 | 360 | 33,000 | | [g]150.0 | do | 1,600 feet inverted siphon, 40 inches diameter; cost $10,000. |
| Do | | | | | | | | | | | | | | | do | |
| Basin Creek | | | | | | | | | | | | | | | do | |
| Glacier Creek | | | | | | | | | | | | | | | do | |
| Dexter Creek | | | | | | | | | | | | | | | do | |
| Cripple River | | | | | | | | | | | | | | | do | |

[a] Not visited.
[b] Bottom boards and side boards.
[c] Posts, sills, and stringers, not sawed.
[d] Data estimated.
[e] Ditch and flume.
[f] See special description of the Miocene ditch (pp. 123-126).
[g] 1901.

TABLE 8.—*Water conduits in Alaska and the North*—Continued.

| Locality. | Flumes. | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Length. | Grade in feet per mile. | Width. | Depth. | Size of boards. | Size of posts and sills. | Size of stringers. | Average amount of water used, miner's inches (1.5 cu. ft. per min.). | Greatest head. | Cost of construction per mile. | Annual cost maintenance per mile. | Cost of lumber per M. | Average open season. | Notes. |
| | | | *Ft. in.* | *Ft. in.* | *Inches.* | *Inches.* | | | *Feet.* | | | | | |
| Council district: | | | | | | | | | | | | | | |
| Ophir Creek | | | | | | | | | | | | | | |
| Do. | 900 feet | 6.60 | 5 8 | 4 0 | 1¼ by 12 | 4 by 6 | | 3,300 | 200 | [a]$20,000 | | $80.00 | June 10–Oct. 10. | There are completed or under construction in Ophir Creek, 55 miles of ditches. |
| Do. | | | | | | | | | | | | | do | |
| Do. | | | | | | | | | | | | | do | |
| Do. | 1,000 feet | 6.60 | 4 0 | 2 4 | 1¼ by 12 | 2 by 4, 6 flat | 8 by 8 inches by 12 feet. | 1,500 | 100 | | | 80.00 | do | 20 piers and trestles, pier cribs rock filled. |
| Do. | | | | | | | | | | | | | | |
| Gold Bottom | | | | | | | | | | | | | June 10–Oct. 10. | |
| Solomon district: | | | | | | | | | | | | | | |
| Solomon River | | | | | | | | | | | | | June 25–Oct. 10. | |
| Do. | 1,000 feet | 8.00 | 10 0 | 3 0 | 1 by 12 | 4 by 4, 4 by 4 | | 1,200 | 120 | | | | do | |
| Port Clarence district: | | | | | | | | | | | | | | |
| Sunset Creek[b] | | | | | | | | | | | | | June 15–Oct. 1. | |
| Klondike River | 2,600 feet | 2.22 | 2 6 | 2 6 | 1¼ by 12 | 4 by 4, 4 by 4 | 4 by 6 | 500 | | 6,000 | | 50.00 | | Estimated capacity, 1,000 miner's inches. |

[a] Data estimated.

[b] Not visited.

Long conduits are used in order that the difference in elevation between the source of the water and the point where it is used may afford the pressure. The length of the conduit will depend on the slope or grade of the surface. In regions of low mountains and gentle slopes, long ditches or flume lines are necessary. In steep mountain regions the same results are attainable with short conduits.

In the gravel-mining districts of Alaska examples of the two extremes are found. It is an unfortunate, but at the same time natural, result of the governing geological conditions, that to obtain the use of water under head ditches must be the largest in the richest and most promising placer districts of Alaska. A long head ditch is expensive both to construct and to maintain. That the expense of such undertakings has not deterred miners from attempting them is evidenced by early California operators, where single companies sometimes built more than 100 miles of ditch line. Each mile of ditch line or other form of conduit adds from $2,000 to $15,000 to the initial capital necessary to start the hydraulicking operation, and increases the annual cost of ditch maintenance.

Some of the precautions which apply equally well to any country regarding the construction of water conduits, found in the text-books on hydraulic mining, may well be repeated here.

Van Wagenen[a] says:

> When the miner has measured the stream from which he is to draw his water supply, and has determined the point where he will tap it, he is prepared to consider the question of water channels. These may be of three kinds—the ditch, the wooden flume, and the iron pipe. * * * It is generally desirable to have the least possible fall in a water channel, or, in other words, to bring the water to as high a point of the ground to be worked as circumstances will allow. As the friction of the sides and bottom of a channel retards the flow, and necessitates a higher grade than would be necessary if there were none, it becomes of importance to decrease this element as much as possible. On this score wood and iron waterways present decided advantages, owing to their comparative smoothness. In any case, however, the quantity of friction developed depends upon the wet perimeter of the channel used. The following law will therefore be found to be of service:
>
> *The least wet perimeter that will hold or carry a given volume is attained when the width of bottom is from one and three-fourths to two and one-fourth times the depth of the sides.*
>
> For example, a channel having a cross section of 510 square inches will develop the least amount of friction when its dimensions are 15 by 34 or 17 by 30, or somewhere between these measurements. A knowledge of this fact will be found serviceable in constructing flumes. The least perimeter, of course, requires the least lumber, and many thousand or million feet may be saved in a long flume by building in the correct proportions.

Bowie[b] says:

> All water courses on the line of the ditch should be secured. Their supply partially counteracts the loss by evaporation, leakage, and absorption, and frequently furnishes an additional quantum of water during several months of the year.

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[a] Van Wagenen, T. F., Manual of Hydraulic Mining, 1880, p. 51.

[b] Bowie, A. J., jr., A Practical Treatise on Hydraulic Mining in California, 1885, p. 135.

> At proper intervals waste gates should be arranged, so as to discharge the water when necessary without risk of damage to the ditch. In regions of heavy snow these wasteways should be provided at intervals not greater than one-half mile. * * * Ditches poorly built in the beginning subsequently require large and constant expenditure, and lose considerable amounts of water

George H. Evans[a] says:

> At the different points along the line of race where fluming has to be resorted to, allowance should be made for an increase in grade, in order that the flume can be constructed of much smaller dimensions than the ditch and yet carry all the water required. * * * It will be well to remember that practical results have demonstrated that in ordinary ground the water should travel at the rate of from 180 to 200 feet per minute. Then the grade will be determined by the dimensions of the ditch and its intended carrying capacity.

The above quotations may be considered as of general application, but many modifications in the hitherto accepted practice of ditch building have been found advisable in Alaska, and will be referred to, especially in the description of the practice in Seward Peninsula. Regarding the capacity of open conduits, Bowie makes the following statement:

> In the mining districts of California ditches are constructed badly, with steep grades and on irregular lines with numerous sharp curves. The cross sections, originally uniform, became more or less varied. Absorption, percolation, evaporation, and leakage reduce the flow. A distinct, reliable factor for each of the sources of loss can not well be incorporated in the coefficient of discharge. * * * The simple formula $Q=ac\sqrt{rs}$ expresses more fitly the result of experience in such cases, wherein:
>
> $Q$ is the quantity of water which the ditch is capable of carrying in cubic feet per second.
>
> $a$, the effective area of cross section of ditch, as originally constructed, in square feet.
>
> $r$, the hydraulic mean depth in feet.
>
> $s$, the fall of surface in a unity of length.
>
> $c$, a coefficient covering all common losses.
>
> Statistics derived from experience on the Milton, La Granger, and Bloomfield ditches in California led to the adoption of values of the coefficient $c$ varying from 31 to 45, in estimating the capacity of ditches in heavy grades of 40 miles' length flowing from 60 to 80 cubic feet per second.
>
> The Texas Creek branch ditch (of the North Bloomfield ditch) is about seven-tenths of a mile long. Its sectional area is 13.5 feet and the grade is 20 feet per mile. The sides are rough and the curves sharp. With a flow of 32.8 cubic feet per second, the ditch runs about full. The value of $c=33$. In connection with this ditch there is a rectangular flume 2.67 feet wide by 2.83 feet deep, made of unplaned boards, set on a grade of 32 feet per mile. The flume has some sharp but regular curves, and the water from the ditch runs nearly full at these points. With the discharge 32.8 cubic feet per second, $c=59$.

Although the losses in leading water in Alaska have not been determined by exact experiments, it is likely that they are less than in California. In the interior the value of $c$ for ditches would be slightly higher than in California, while in the ditches made on gentle slopes in Seward Peninsula *c* would be considerably higher. Its value can not be known until accurate experiments have been conducted on the various ditch lines.

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[a] Evans, G. H., Practical Notes on Hydraulic Mining. San Francisco, 1898.

Examples of the discharge of ditches in California, given by Bowie, are as follows:[a]

| Locality. | Length. | Grade. | Size. | Discharge. |
|---|---|---|---|---|
| | *Miles.* | | | *Miner's inches.* |
| North Bloomfield... | 55 | 16 feet per mile.... | 8.65 feet top by 5 feet bottom by 3½ feet deep. | 3,200 |
| Dutch Flat.......... | 13 | 13½ feet per mile... | 6½ feet top by 4 feet deep. | 3,150 |
| La Grange........... | 20 | 7 to 8 feet per mile. | 9 feet top by 6 feet bottom by 4 feet deep. | 2,400 |

In considering ditches and ditch building, the three provinces of Alaska, namely, the South Coast, Interior, and Seward Peninsula, will be taken up in succession.

### SOUTH COAST PROVINCE.

In the South Coast province of Alaska the topography permits a rapid increase of head in short distances. (See Pl. XVI, *A*, p. 114.) The ditch line of the Alaska-Treadwell Gold Mining Company, 14 miles in length, delivers water from the penstocks above the mill—to which it supplies power—at an elevation of 480 feet. As may be seen from the map (fig. 19), the ditch heads in Fish Creek 1,000 feet above the sea. The water at the upper penstock, midway on the ditch between Douglas and Treadwell, is taken out at an elevation of 600 feet above the sea. The distance from the penstock to the nozzle is only 1,500 feet on slope, and it may easily be seen what advantage is gained from the steep topography. The grade of the mountain, nearly 30 feet in 100, is such that were a sufficient amount of water available in one of the near-by creeks no ditch at all would be required. The water could be led out by a few hundred feet of flume to a penstock, and a pipe line led directly from this to the mill. The Treadwell ditch line is composite, so far as its contained water is concerned, since it taps successively Fish, Eagle, Cowee, Lawson, Paris, Bullion, and Ready Bullion creeks. Its length is not extreme considering the amount of water obtained, but it is considerably greater than would have been necessary if all the water could have been obtained from a single creek or river.

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[a] Bowie, op. cit., p. 139.

The apparent extraordinary fall of the Treadwell ditch (over 20 feet to the mile) may be accounted for by the fact that it has, at each creek which it taps in its course, a drop off of 6 to 7 feet, and reservoir, in which are relief gates, as illustrated in the photograph of the ditch at Lawson Creek. (See Pl. XVI, *B*.) One of the waste gates shown on Pl. XVI, *B*, is 5 feet 4 inches wide; the other is 6 feet 7 inches wide and 3 feet 3 inches deep. There is a smaller auxiliary gate 4 feet 6 inches wide and 2 feet deep.

The Treadwell main ditch is 14 miles in length, and is supplemented by 4 miles of subsidiary ditches.[a] The main ditch heads in Fish Creek, and its supply of water is obtained successively from Eagle, Cowee, Lawson, and Paris creeks, while an additional ditch brings water from Ready Bullion and Bullion creeks. In summer the water is used for running 760 of the 880 stamps of the mills, but in winter it is not used for power purposes.[b]

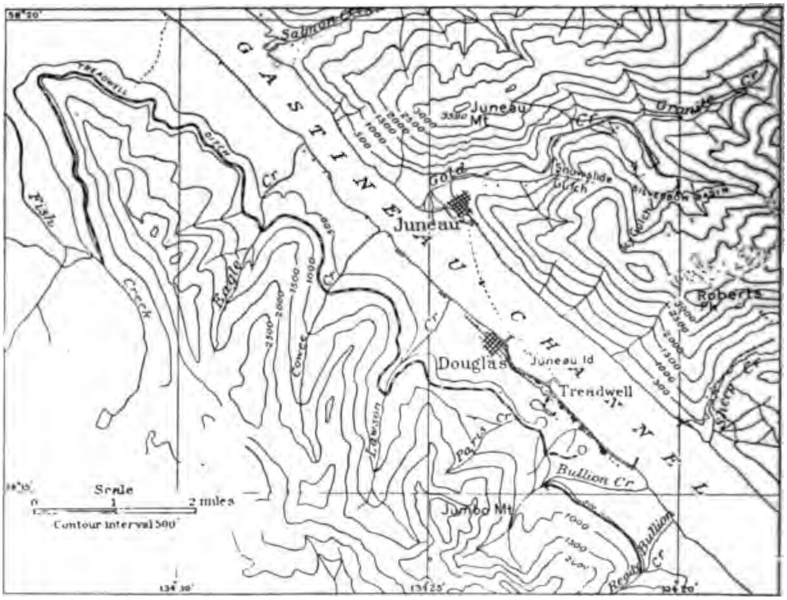

FIG. 19.—Map of portion of Juneau district.

The main ditch has a width of 7 feet on top and 6 feet on the bottom, and a depth of 5 feet. In 1904 construction was going on in portions of the ditch line as follows: Twelve-inch posts, in sets with 5-foot centers, were being placed along the inside of the ditch, leaving space 5 feet in the clear at bottom and 4½ feet in the clear at top, with 7-inch cap logs 4½ feet between shoulders. Lagging split 3 inches thick and 1½ feet wide was laid inside the posts. Moss and sod were

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[a] See Kinzie, R. A., The Treadwell group of mines, Douglas Island, Alaska: Trans. Am. Inst. Min. Eng., vol. 34, 1904, p. 374.

[b] *It is to be regretted that full data regarding the Treadwell ditch are not at hand.*

*A.* TOPOGRAPHY, SOUTH COAST PROVINCE.

*B.* TREADWELL DITCH LINE AT LAWSON CREEK, DOUGLAS ISLAND.

then filled in between the lagging and the side of the ditch, poles laid lengthwise over the caps, and sod laid over the whole. Fluming has been found very unsatisfactory at the Treadwell. The flume was banked with sod on sides and top. Anchor ice in winter collected in the bottom of the flume, and was removed with great difficulty.

The Treadwell ditch for long distances is bridged with poles and brush, which in many places are covered with sods. Sodding is said to be the best preventive against freezing, not only here, but in all parts of Alaska.

The pressure boxes are connected with the ditch by flumes 35 feet long, 5 feet wide, and 3 feet deep. The water first enters a gravel tank which is cleaned twice a year. This tank is built of 2-inch lumber and is 8 feet wide by 9 feet long and 14 feet deep. From the tank the water flows through a 4-foot section of flume 3 feet 3 inches wide, the bottom of which is only 5 feet below the top of the tank, into the main penstock. This is 11 feet long by 9½ feet wide, inside measurement, and 14 feet deep. It is built of 3-inch lumber, with 8- by 10-inch posts, sills, and caps, with 4-foot centers. The penstock is fitted with a wooden inclined grizzly to catch any leaves or refuse that pass the gravel trap. The penstock and sand trap combined make a structure 20 feet 6 inches long by 10 feet 10 inches wide by 14 feet deep, outside measurement. The penstock, as illustrated in Pl. XVII, *A*, is banked halfway up the sides with sod, and the pipe line leading from it is covered with sod. The sand trap is provided with a sluicing-out gate on the side opposite that shown by the photograph.[a]

Pl. XVII, *B*, illustrates a method of guying the pressure box on steep slope, in use by the American Gold Mining Company in Silver Bow basin, Alaska. The intake of the pipe line is protected by shed from heavy snows.

As water is in use for placer mining only during the open season, few instances were seen in Alaska of attempts to sod up the ditches and connections to prevent freezing in winter. At the Treadwell, however, the water is used for generating power. Pipe lines, especially in Seward Peninsula, are frequently sodded over, as much for protection against rust as to prevent freezing.

In Silver Bow basin, east of Juneau, ditching has been found impracticable and all conduits are flumed. In general in southeastern Alaska fluming will be found cheaper than ditching, as ditches would have to be cut in solid rock for the most part, and in numerous places the mountain slopes are so steep as to render their construction impossible.

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[a] See Bowie, A. J., jr., A Practical Treatise on Hydraulic Mining in California, 1885, p. 177, for plan and elevation of North Bloomfield pressure box.

The flume of the Jualpa Company,[a] one-half mile northeast of Juneau, is shown on Pl. XVIII, *A*. The water is taken from Gold Creek, the head gate being 9 by 8 feet. The flume is 4,600 feet in length, and has a grade of 1¼ inches in 16 feet, or 34.3 feet to the mile. It is 4 feet 8 inches square, inside measure; the bottom and side boards are 2 inches thick, planed inside, and battened with 3- by ½-inch lumber; there are 4 collars to a box of 12 feet; each alternate collar consisting of a 5- by 5-inch sill, 8 feet long; two posts 4 by 6 inches by 5½ feet high, with a gain of one-half inch into sill, and two caps of 2- by 4-inch pieces nailed to the sides of the posts above the cover boards. The yokes are braced with 2 by 12's. These collars have 6-foot centers, and alternately between them are yokes consisting of a 3- by 5-inch sill, and two 4- by 4-inch posts, uncapped. The flume is covered with boards 1½ by 12 inches by 5 feet, laid crosswise, and nailed. Each box contains approximately 425 board feet of lumber. The cost of the flume is said to have been $2.20 per foot, or $11,616 per mile, including lumber and labor. This did not include the cost of shooting rock for the flumeway, which was as high as $5 per foot in places. A head of 225 feet is obtained, and the capacity of the flume is said to be 5,000 miner's inches.

The pressure box to which the flume leads is 12 by 18 feet by 11 feet deep. It is built of 1½-inch lumber, and has collars of 8- by 8-inch timber. The pipe at the intake end is 36 inches in diameter.

In building flumes about Juneau trestling frequently has to be resorted to, thus adding greatly to the expense. In general it may be said that the cost of building a 4-foot flume in the South Coast province will not fall greatly below $10,000 per mile, and will occasionally be double this amount.

For small flumes, Bowie gives the following specifications:[b]

*Specifications for flume 2½ feet wide, 2½ feet deep; 12-foot box.*

| | Feet. |
|---|---|
| 3 caps, 4 feet by 3 by 4 inches | 12½ |
| 6 posts, 3 feet by 3 by 4 inches | 18 |
| 9 planks, 12 feet by 1½ by 6 and 12 inches | 135 |
| 3 sills, 4½ feet by 4 by 4 inches | 18 |
| 2 stringers, 12 feet by 4 by 6 inches | 48 |
| 6 battens, 12 feet by 3 inches by 1 inch | 14 |
| 1-foot plank, 12 feet by 10 by 1½ inches | 15 |
| Total lumber in one box | 264½ |

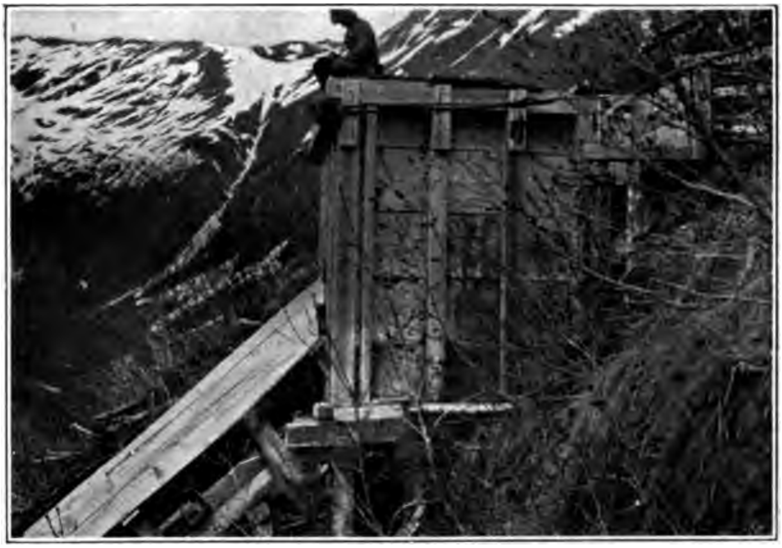
Number of boxes per mile, 440.

---

[a] Recent information (Mining and Scientific Press, December 31, 1904), states that elaborate preparations for operating this property were made the last summer. These include the interception of the water of Gold Creek above the property by a dam, and its diversion to a flume to carry it past the hydraulic pit. The flume is 20 feet wide by 9 feet deep, and 4,250 feet long, 2,250 feet being trestled. It contains 1,200,000 feet of lumber, 6,000 linear feet of hewed timber, and 20 tons of nails and bolts.

[b] *Op. cit.*, p. 119.

*A*. PRESSURE BOX IN TREADWELL DITCH.



*B*. METHOD OF STAYING-PRESSURE BOX, SILVER BOW BASIN.

Reckoning the cost of lumber for a box at $4 and the labor of 1 man at $5 for one day to each box, such a flume would cost $4,000 per mile, exclusive of survey, rock work, and trestles. It is difficult to see how the cost of building even small flumes can fall below $7,000 per mile in the South Coast province of Alaska.

### INTERIOR.

In the interior Yukon and Tanana fields ditching is practicable and is recommended in preference to flumes. There are difficulties, but they are not insurmountable. In the interior no large ditch enterprise similar to those of Seward Peninsula has been undertaken. The Atlin field has afforded data regarding building of ditches, but conditions are hardly comparable with those in the Yukon region. With the exception of the small amount of bench mining in the interior only a small amount of gravel is handled daily, and ordinarily water under very low heads is used.

The future of hydraulic mining in the Klondike, Birch Creek, Fortymile, Eagle, and Tanana mining districts is not promising. The gentle slopes of the mountains, the low grade of the creek valleys, and the sparse amount of water are unfavorable to obtaining a sufficient amount of water for hydraulicking at a working head. A peculiar drawback, depending on the geological history of the region, is the fact that the level tops of the Yukon and Tanana mountains represent a partially eroded peneplain of very wide extent—300 miles from east to west and 200 from north to south, approximately. This area, in which the placer fields of the Yukon-Tanana rectangle are situated, is, as it were, a table-land, the top of which tilts slightly to the south, and into which the modern streams have cut, rounding off the residual parts between them into low dome-shaped mountains, averaging 3,000 feet in height, and varying from 6,000 to 2,000 feet.

A comparison of the topography of the South Coast province with that of the interior is illustrated by Pls. XVIII, *B*, and XIX, *A*. In the interior there are no sharp declivities, no waterfalls, no sudden descents in the surface. The grades of creek valleys are seldom over 3 feet in 100 and are commonly 1 foot in 100. (See table 8, p. 104.) If 2 per cent is taken as an average grade and it is assumed that a ditch must be three times as long as the creek on which it is to furnish water under head at a lower point, it is evident that only 300 feet head would be obtained by 10 miles of ditch. Such a ditch taken from one stream would afford only 200 miner's inches of water, a fair average for the region during the four months of the summer season.

Within leading distance there are no high mountains from which to draw a supply. The alpine peaks of the Alaska Range lie to the south of the Tanana, and could not by any possibility be made to afford water to be used for hydraulicking. (See Pl. XLI, *B*.)

Ditches and flumes for conveying water must, as hitherto, be of small size. They will deliver a small amount of water at a low head, 100 miner's inches of water at 75-foot head being about what may be obtained under favorable conditions.

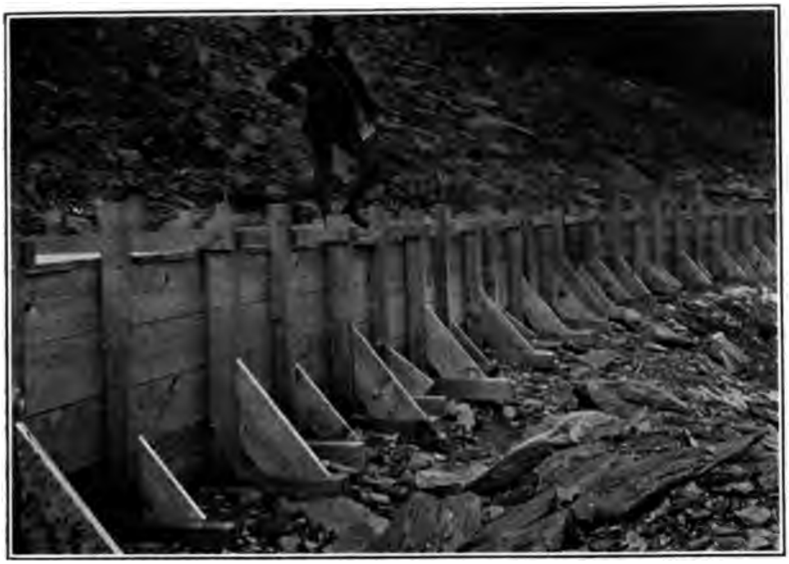
Special difficulties encountered in the interior of Alaska and the North generally have been found to be as follows:

At Atlin the action of frost causes the ditch to enlarge after construction. The opinion was expressed that steel-pipe conduits do not freeze as quickly as flumes. In a 19- by 40-inch flume of 1,960 feet in length it was found that the frost heaved the flume and put it out of grade each season, but even then it was regarded as more economical than a ditch. The flumes on McKee Creek have to be gone over every spring to restore the grade. Sills are set in 4 inches of blocking on the stringers so that the grade can be regulated. It was also said that the native Atlin lumber is preferred for the flumes and sluices, as imported Pacific coast lumber warped badly. If native lumber is used, 8 inches is the greatest width available. Notwithstanding the cost of maintenance, ditches are much used at Atlin, one company operating 10 miles of ditch.

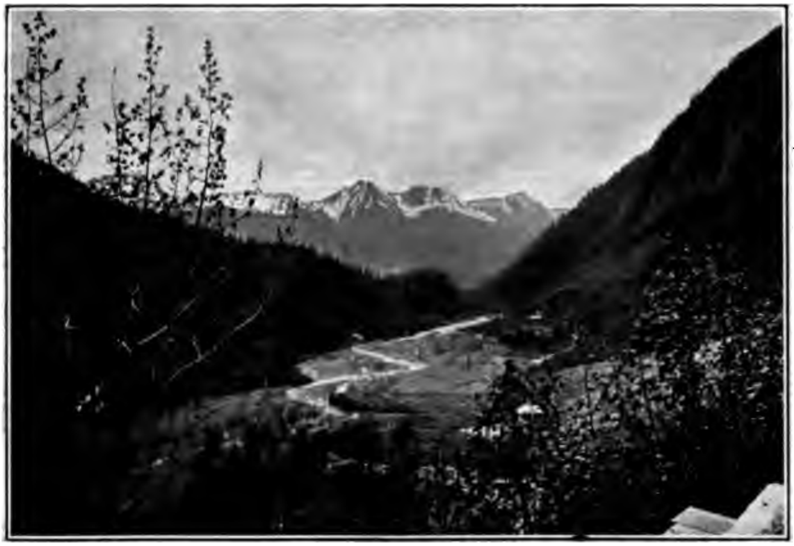
In the Klondike district, where nearly all ground is solidly and permanently frozen, it is said that three years after construction must be allowed to get a ditch into condition to stand. Various estimates have been made and projects formulated in this field for bringing in water from a distance for the purpose of working the remaining gravels of the benches (the so-called "White Channel") which lie at an average elevation of 270 feet above the rich bottoms of Bonanza, Eldorado, and Hunker creeks. The building and maintenance of a ditch having its source in some of the higher tributaries of Klondike River and affording water at a sufficient head is not regarded as an impossible engineering feat, but up to the present the cost has been considered prohibitive. Dry seasons alternate with wet ones, and in consequence the amount of water available during a given season might vary from, say, 1,000 inches to 5,000 inches. No calculations based on the full capacity of the ditch could be made in advance, therefore, regarding the season's product.

Recent information concerning the Acklen ditch, led along the north side of Klondike River near Dawson for the purpose of hydraulicking high benches bordering that stream, has been embodied in the table. It is stated that the cost of the earth excavation of this ditch, 7 by 4 by 2½ feet deep and 41,500 feet in length, was $34,000.

Small storage reservoirs are used by many of the Klondike operators to impound the small and variable amount of water available for working the benches of the White Channel. On Hunker Creek a small ditch 3 by 2 feet and 4 miles long is in excellent condition after



*A*. FLUME OF JUALPA MINING COMPANY, SILVER BOW BASIN.



*B*. TYPICAL SOUTH COAST TOPOGRAPHY, GOLD CREEK.

four years' service. The annual cost of maintenance is $1,000, or 10 per cent of the original cost.

In ditching in bad ground it has been found beneficial not to remove the moss carpet, but to cut it in 4-foot lengths, transverse to the line of ditch, then to roll it back along the lower edge, and after the ditch is dug to let it drop so as to cover the side of the ditch.

In the region adjacent to Eagle, Alaska, it has been found that where the topography did not admit of ditching the maintenance of flumes is very expensive and that the use of steel-pipe conduits is preferable.

A useful suggestion for flume building was obtained on Deadwood Creek, in the Birch Creek district. Flat timbers obtained in the neighborhood were used, and the flume, 3 feet by 1 foot, was calked with moss inside. The cost was only 25 cents a foot, and the flume is said to be good for ten years. In this locality it is almost impossible to secure sawed lumber at any price unless the operators whipsaw it themselves. In all northern latitudes the moss is an excellent calking material.

Through the interior country it has been found that ditches on the south slopes of the mountains can be made with much more success than on the north slopes. In fact, the north slopes are frequently bare rock, while on the south slopes a good deposit of earth has accumulated.

The presence of crystosphenes,[a] which are found on the slopes as well as in the creek valleys in all parts of the north where perpetual frost prevails, is the ditch maker's greatest obstacle. Mr. Tyrrell says: "As a rule they [the crystosphenes, or so-called 'glaciers'] occur as more or less horizontal sheets of clear ice, from 6 inches to 3 feet in thickness, lying between layers of 'muck' or fine alluvium, usually where the 'muck' is divided horizontally by a thin bed of silt or sand; and most of them, as far as my observation goes, are from 2 to 4 feet below the surface, though some are deeper. They approximate closely to the slope of the surface, under which they lie." These sheets are from 25 to 150 feet in diameter, generally somewhat oblong. They are explained by Mr. Tyrrell as due to the freezing of seepage water, causing a gradual accretion to the mass annually. Mr. I. P. Tolmachof[b] has examined similar occurrences in northeast Siberia, and explains them as fossil snowbanks. Whatever their explanation, they are exceedingly common both in the interior and in Seward Peninsula. They are found not only in creek beds, but on the slopes, and show no regularity in distribution.

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[a] Tyrrell, J. B., Crystosphenes, or buried sheets of ice, in the tundra of northern America: Jour. Geol., vol. 12, 1904.

[b] Ground ice of Berezovka River: Proc. R. Russ. Min. Soc., St. Petersburg, 1903.

Where such a mass of ice is met sod and moss must be extensively used, and both the first construction and annual maintenance will be much greater than in other portions of the ditch. For an account of the method of dealing with such ice sheets see the description of ditch building in Seward Peninsula.

In the Fairbanks district so little ditching has been done that no results gained from experience were available. The frozen ground has, however, been found so bad for ditching that the cost of annual maintenance is from one-fourth to one-half as much as the first cost of construction. Such expedients as patching with canvas have been adopted as a temporary makeshift. The small ditches which have been constructed are run with as flat a grade as possible to avoid cutting out. Here as elsewhere in the interior the use of sod for lining the sides of the ditches can not be too highly recommended.

Small reservoirs in the creek beds for impounding sluice water are in use and will be described in connection with open-cut mining.

### SEWARD PENINSULA.

The following notes regarding the building of ditches have been contributed by Mr. Guy A. R. Lewington, of Nome, Alaska, manager of the mining properties belonging to the North American Transportation and Trading Company:

> Ditch building in Seward Peninsula has become one of the most serious questions with reference to economical mining and the general stability of the whole mining industry. Water under pressure for hydraulic mining is the all-important condition, and to this end I submit the following, which may be of some interest to those contemplating investment in this section.
>
> The necessary equipment consists of plows, scrapers, and graders of the usual kind in use in the States, and, of course, a camp outfit of tents, both for living and stable purposes.
>
> The conditions to be contended with in this country are different in almost all respects from those which have come under my observation elsewhere. The most serious of these is the building of a tight and strong ditch over "glacier,"[a] of which much is encountered wherever ditches have been constructed. Under this condition it has been found best to dig as shallow and broad a ditch as possible, not digging much below the moss, and to build the outer bank of the ditch up to the required height by the use of the sod which covers the whole country. This sod in a short while settles and knits itself together, and thus becomes a very serviceable bank. It will not cut or wear out, and the older it gets the better it becomes. When, however, it becomes evident that the bottom of the ditch is cutting and wearing away, sod again must be resorted to, and by lining the bottom of the ditch with it the trouble may soon be overcome. In this way a ditch can be made over perpetually frozen ground where otherwise it would be impossible. Much ditch has to be constructed over loose stones with little or no sediment between them. In this case the ditch must be lined with sod and all holes must be filled by tamping sod into them as far as possible. This being done it will be found that the water traveling through the

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[a] The buried ice sheets.

*A*. TYPICAL INTERIOR TOPOGRAPHY, MASTODON CREEK, BIRCH CREEK DISTRICT.

*B*. DITCH BUILDING, HEAD OF NOME RIVER, SEWARD PENINSULA.

ditch will deposit sediment over the sod and level it up, and that after a little while it will become tight. Puddling, of course, will help any ditch here or elsewhere. Those two conditions are the only serious ones to be contended with in ditching ground.

A strong plow drawn by 2-horse team is the first implement used. (See Pl. XIX, *B*, p. 120.)

All being ready, the driver is instructed to plow a single furrow, following as closely as possible from one survey peg to the next, following the natural contour of the country. This he does for say a distance of one-half mile, thus establishing the ditch line. The plowing is continued to a width sufficient so that, allowing plenty of slope for the inner bank, the required depth of ditch may be obtained. The grader is next used for the purpose of removing what has been plowed to the outer bank of the ditch. This being done the ditch will look much like a wagon road. Then the plow is used again, plowing as before a single furrow, following as nearly as possible the first furrow plowed, which is plainly visible. This second plowing being done, the scraper is resorted to, and the loose plowed material is scraped from the ditch to the outer bank, building it up. This work is repeated until the ditch is almost completed. All that remains to make an excellent ditch is to level up the bottom and to slope the ditch to required dimensions. This work is done by hand with pick and shovel. The plow and scraper should do almost all the work, however, so that as little as possible remains to be done by hand. After completion of the ditch only a small head of water should be allowed to flow through it for a few days, until it has become well soaked; then the head may be gradually increased a little daily until the full capacity is reached.

All water should be turned out of the ditch before the freeze-up in the fall, and the ditch made as dry as possible by the opening of all waste gates, of which there should be one at least every one-third of a mile. These waste gates should be left open, to enable the water during the spring thaw to run out of them instead of filling the ditch with water and overflowing its banks. These waste gates should be cleared of all snow and ice at the first approach of a thaw in the spring, in order that the water may have a free outlet. This is very important. In the spring no water should be allowed to run through the ditch until at least 2 to 3 inches in depth of the ditch has thawed, and then only a small head to start with, as frozen ground cuts very rapidly. If this work is carefully done I have no doubt that the ditch will be ready for work by the time it becomes possible to mine.

Considering the remoteness of the country and its high latitude, ditch building can be done at a cost surprisingly low. The entire absence of timber, small amount of rock work, generally good soil, and gentle slopes of the hillsides are conditions which make ditch building very feasible in the auriferous portions of Seward Peninsula.

Fluming should not be employed unless absolutely unavoidable, as frost and snow in winter play havoc with flumes, and the swelling and contracting of the ground, due to alternate freezing and thawing, continually keep the flume off grade. There are few instances where it is necessary to use a flume.

Much of the ditch construction at Nome is now done by contract. The following information was supplied for this report by one of the ditch contractors at Nome:

In the construction of ditches on Seward Peninsula the following three types of machines are used, all being drawn by horses: The ordinary road grader, the horse scrapers, which are so extensively used in California, and an ordinary breaking plow.

After the ditch line has been surveyed and staked 30 men and 20 horses, for example, with plows and scrapers, are put to work. The plows proceed ahead of the grader, and the cutting is continued until the uphill side of the ditch bottom is nearly cut to grade. At this stage, which is represented ideally by fig. 20, it will be seen that the contents of the cut A, B, C has been thrown up to form the bank C, D, E on the outer side of the proposed ditch. The remaining work consists in removing the earth in the portion B, F, H, which is partially uncut and partially built up by the grader. The most economical work is done when the portion removed by the grader is so balanced as to demand the least cutting by the scrapers in throwing up the necessary bank. It will be seen from the above statements that the burden of the work falls on the first plows and on the grader, though the subsequent work of the scrapers and hand finishing is just as important.

Four horses and 1 man are usually used to a grader, though 8 horses are often used. The scrapers use from 2 to 4 horses, depending upon the size of the ditch. Four horses are not generally used to a scraper unless the ditch is to be 7 feet or more wide. The plows breaking first ground usually need 4 horses each. It may be said that where a slope is steep very little or no cutting is needed on the lower side of the ditch, since the dirt necessary for the bank can all be obtained in reducing the upper side to grade.

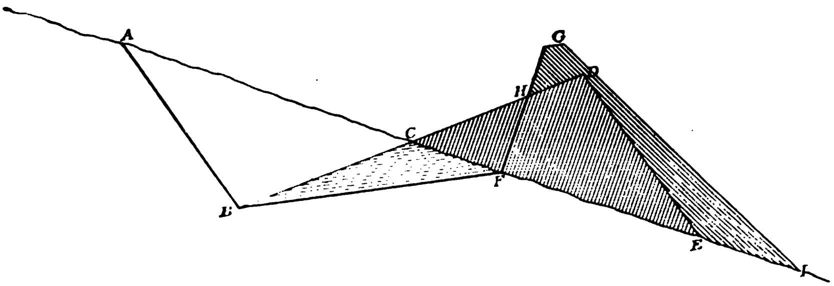

FIG. 20.—Section of ditch in construction.

Special methods are necessary when the ditch passes through sections underlain by ground ice, as previously described, or runs over sections of rock. Very careful work is needed when the rock is broken and fractured. It has been found bad practice to cut through the stringy moss which overlies the masses of ground ice, generally referred to as "glacier;" in fact it is disastrous to the permanency of that section of the ditch, and is the beginning of never-ending repairs, since the ice continues to thaw, causing constant leakage. The best practice is to build sod walls on the lower side, leaving the moss undisturbed. All rock work must be done by hand, and where the ditch passes through fractured material all cracks must be filled with moss. Too much care can not be observed in the latter detail, and, especially during the first weeks of use, men must be kept constantly traversing and repairing those sections where leaks are apt to occur. The stirring up of the water by men walking along the bottom of the ditch is a good practice in the early stages, for silt, in addition to the sod, is a most valuable factor in filling the cracks.

A scraper will work to great advantage in decayed schist, which needs no lining, as it holds water better than any other ground encountered and cuts out less. Fluming does not pay when there is a possibility of ditch building. In fact, it has been often stated by men familiar with ditch construction, that where possible, it is profitable both as regards first cost and subsequent maintenance to build a ditch in place of fluming, even if the distance necessary to be covered by the former be ten times that of the latter. Many slopes apparently not permitting a ditch cut, owing to the presence of broken rock and talus slides, on close examination are found to be favorable, for if 2 or 3 feet of this loose material is moved there are excellent opportunities for comparatively cheap rock cuts.

When, however, it is deemed impracticable to construct a ditch, and where a flume must be built crossing a gully, a very efficient foundation can be made by digging shallow holes, filling with gravel, and placing on top a wide plank to distribute the load. If the trestle rests on such foundations, and the underlying ice is not disturbed, much trouble from settling will be avoided. The following are a few of the costs representative of ditching in various materials:

*Cost of ditching in various materials.*

| | |
|---|---|
| Soft muck and tundra, per cubic yard | $0.75 |
| Gravelly dirt, per cubic yard | .65 |
| Decayed schist, per cubic yard | .40 to .60 |
| Rock work, fairly solid, per foot | 1.75 |
| Schist in place, per cubic yard | 1.00 |
| Loose rock, per cubic yard | 1.25 |

Ditching in muck which heaves is very expensive, and no general figure can be given.

A ditch carrying 1,000 miner's inches will cost, under fair conditions, $2,000 per mile. One with the capacity of 4,000 miner's inches will cost between $4,000 and $5,000 per mile. Though much affected by varying local conditions a conservative estimate for general work is $1 per cubic yard throughout.

The following account of the construction of the extensive water conduits built by the Miocene Ditch Company was obtained from Mr. J. W. Davidson, the company's consulting engineer.

The water is taken from Nome River and its tributaries for the most part, although a small portion is taken from Snake River. At the intake near the head of Nome River the elevation is 572 feet.[a] At Hobson Creek a large head dam has been constructed entirely of sod. The dam was started at bed rock and is 20 feet in width by 130 feet in length and is provided with a waste gate 10 feet wide and 10 feet deep in the center. It has been found entirely satisfactory. The ditch, with its branches and laterals, is 54 miles in length. Of this, the main ditch is 31¼ miles from Hobson Creek to the tunnel. This ditch, as far as the "X," was made 10 feet on the bottom, 14 feet on top, with a depth of 3 feet. It has a grade of 3.37 feet to the mile. Seventeen miles of ditch were made from the head of Nome River to Hobson Creek, with the dimensions 8 feet wide on the bottom, 11 feet on the top, and 3 feet in depth, with a grade of 4.5 feet to the mile. From the "X" to the tunnel, as represented on the map, the ditch has the same dimensions as the upper end, and a grade of 6.5 feet to the mile. The ditch was constructed to carry 3,000 miner's inches of water below Hobson Creek, and at the upper end to carry 2,500 miner's inches. The actual amount of water available under average conditions is said to be 2,000 inches. The ditch was begun on July 6, 1901, at the "X." (See fig. 21.) The following year 75 horses and the necessary equipment—

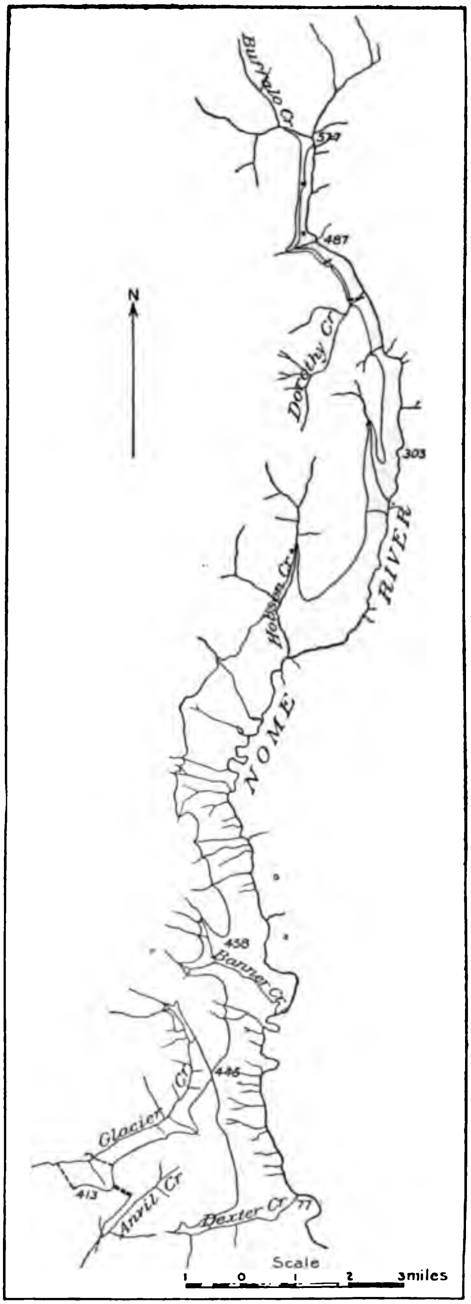

FIG. 21.—Map of Miocene ditch.

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[a] According to levels by U. S. Geol. Survey.

*A.* MIOCENE DITCH, GLACIER CREEK.

*B.* FLUME OF HOT AIR COMPANY, OPHIR CREEK.

plows, graders, and scrapers—were brought in and the ditch was completed to Hobson Creek. In 1903 the ditch was completed to the head of Nome River and a branch ditch was constructed to tap Snake River. Sixty-eight days were consumed in building the ditch from Hobson Creek to Dexter Creek, forty-eight days from Hobson Creek to Nome River. This latter figure includes the enlargement of the originally built Hobson ditch and the building of the Snake River ditch.

The method of construction has been described above. A plow was used for breaking the furrows and the grader was used for bringing the cut to grade, 4 horses being used to a team. The ditch making was carried on at the rate of one-half mile per day, taking the triangular section off down to a level. This operation was followed by a second plowing. The grade stakes were set at a certain distance down the hill from the lower bank, and the men were directed to push the dirt so as not to reach beyond the line of stakes. From 60 to 70 men were employed all the time, and the number of horses varied from 50 to 100. The cost was reckoned at $400 for each horse for a period of eight months, and the wage of a man was reckoned at $7.50 per day. Two 4-horse teams were employed all the time to haul feed. (See Pl. XX, A.)

The grading gang consisted of one 4-horse breaking plow and a grader, with 2 men and 8 horses. A scraping gang consisted of 2 plows and 9 scrapers. There are thus 11 drivers, 2 men plowing with 2-horse plows, 9 scraper men, and a foreman, beside the grader. The general cost of the smaller ditch is given at $2,300 per mile. It is to be noted that in this construction the upper bank was not sloped at all, as it was found that this sloping is very little use in the northern regions, the soil standing very well at various angles.

The difficulties with ground ice were very great. At one place 800 feet of such an ice sheet was found, and here the cost of maintenance is exceedingly high. The only way to maintain the ditch is to haul clay down the ditch in boats and dump it in. It is found that if sufficient clay is dumped on top of the ice it stops thawing, but this operation has to be annually repeated. At another point 1,100 feet of flume (8 feet by 33 inches, with double grade) were built over an ice sheet, and so far the ground has settled very little. One and one-half inch lumber is used in construction, at a cost of $200 per thousand feet. For each mud sill cuts were made into the ice from 2½ to 3 feet. After the sills were in position the moss and tundra were rammed back over the ice. The cost of construction of this flume was $1,500, exclusive of the lumber. It is stated that the construction cost much more than it would at present.

At another point, across Manila Creek, an inverted siphon 1,000 feet in length, of hydraulic riveted steel pipe, 40 inches in diameter, 14 gage, is used for 1,000 feet, at a cost of $10,000. It has a dip of

150 feet in the center, and the difference between the ends is 4 feet vertically. The allowance for friction made in this siphon was for 3,000 miner's inches of water. At the upper end the water discharges from a penstock 12 by 12 by 14 feet. Another inverted siphon line 300 feet in length, of 24-inch, 14-gage, riveted steel pipe, is laid across Dorothy Creek.

Around Cape Horn there was a considerable stretch of rock work, 1,300 feet in marble. This had to be blasted with powder, and cost $12,000. There are 5 miles of rock work in all along the ditch line. Between Snow Gulch and Glacier Creek a tunnel 1,800 feet in length has been run through the divide to carry the ditch water. Its dimensions are 4 by 6 feet, and it is timbered in places. The top of the tunnel is level with the top of the water on the entrance side, the bottom of the tunnel being run at a low level so as to completely fill the tunnel. All the rock work in the tunnel was done by hand drilling, and in the winter. The rock was found to be frozen 90 feet vertically below the surface. The elevation of the penstock for "No. 1 Below Discovery," Glacier Creek, where the bulk of the water was used in 1904, is 413 feet, the surface of the water being about 2 feet lower, the available head at No. 1, below Glacier, being 330 feet. The greatest head of water attainable by this ditch on ground now being worked is said to be 360 feet on No. 2, below Glacier Creek.

The complete cost of the ditch and all accessories, including maintenance for four years, is stated to be upward of $300,000. It is the practice to keep 15 men on the whole length of the ditch system during the four months of the working season. This company, besides the use which it makes of the water for its own mining operations, sells some of its water to the miners on Glacier, Anvil, and Dexter creeks at the rate of $1 per miner's inch under pressure, and at 50 cents per inch for water that has been once used.[a] According to measurements made with a Price current meter August 23, 1904, the Miocene ditch afforded 1,074 miner's inches above the No. 1, below penstock on Glacier Creek, and 1,752 miner's inches at a point on Glacier Creek above the tunnel leading a portion of the water to Anvil Creek, the inch equal to 1.5 cubic feet per minute.

In the whole of Seward Peninsula there are approximately 175 miles of water conduits, for the most part ditches actually constructed, and fully 100 miles more are reported as under construction or in contemplation. It is safe to say that the work has averaged in cost over $4,000 to the mile, although it is undeniable that with the advantage of experience the operators can build their ditches for less money in future.

Ophir Creek, in the Council district, has been the scene of the greatest activity in ditch construction, as may be seen by the table. There

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[a] *The miner's inch in this case is reckoned as equivalent to 1.2 cubic feet per minute.*

*A* PIER FOR TRESTLE, HOT AIR DITCH LINE, OPHIR CREEK.

*B*. SMALL STORAGE DAM, BONANZA CREEK, KLONDIKE.

is estimated to be a total of 55 miles of ditches in this basin, including the ditch which taps Pargon River, across the divide. The largest ditch on this creek carries from 1,200 to 3,000 miner's inches, and is through rock and earth for over 18 miles. It was commenced in August, 1901, and work has been done on it at intervals up to the beginning of the season of 1904. It is estimated by the Wild Goose Mining and Trading Company that on Ophir Creek, in ordinary ground, a mile of ditch a week of this size, 16 by 10 by 3 feet, can be built with 32 head of horses and 70 men. In the 5 miles of rock work which were made along the line black powder was used for blasting. The sod walls were found preferable to rock walls in this ditch, as in the one built by the Miocene Company. The water is used for working several claims, a portion being distributed to each; the head attained varies from 170 to 200 feet in the different operations.

In the construction of the Hot Air Company's ditch line on Ophir Creek several long trestles (see Pl. XX, *B*) were found necessary where the ditch crossed and recrossed the many meanders of the creek. It was necessary to sink the posts of the trestling in rock-filled cribs or piers in the creek bed, as shown in Pl. XXI, *A*.

The use of inverted siphons is common in many of the Alaska ditch lines, but presents nothing new over the California practice. The ordinary rules to be observed in conducting water under pressure in pipes, the use of large-diameter pipe to prevent excessive friction, the calculation of the pressure, resistance, the loss of head, and the distribution of air valves are as important as in the case of all pipe lines.

## STORAGE RESERVOIRS.

Lack of water and of sites at a sufficient elevation to afford head render the building of extensive storage reservoirs in Alaska impracticable. The broad, flat valleys, frozen soil, impervious schist bed rock, and the excellent sod material for dams offer conditions which appear attractive. On the other hand, the small and variable amount of rain and snow (see table 2, p. 48) makes it certain that the expensive surveys and construction necessary for reservoirs whose capacity runs into the millions of cubic feet would never be justified by results.

Small storage reservoirs have been successfully built and have given satisfactory results. The capacity of one built by the Anglo-Klondike Mining Company is 400,000 cubic feet, or sufficient to supply 300 miner's inches for fifteen hours for hydraulicking at a head of 150 feet. The dam is 15 feet high and the reservoir is filled from the supply ditch in from a few hours to three days, according to the rains, which are exceedingly variable. This reservoir, which is built at the elevation of the top of the ancient base-level above the highest gravels, is representative of the best that can be accomplished in this line in the interior

Yukon-Tanana field. Mr. J. P. Hutchins[a] states that during the summer of 1904 a reservoir was built in the Klondike which has a capacity of 26,000,000 gallons and requires a dam 40 feet in height.

Many small reservoirs (see Pl. XXI, *B*) are used in the North, affording from 50 to 100 inches of water for a run of a few hours. Reservoirs for the providing of sluicing water for winter dumps have been built on Anvil Creek for catching snow water. In some cases even snow fences have been erected behind the dams to increase the size of the snowdrifts.

The Alaskan miner has at hand the vegetable sod or peat, a material which has proved efficient under Alaska conditions for the building of small dams. The construction of dams is discussed on pages 56–57. In building small storage reservoirs for hydraulicking, it is as necessary in Alaska as elsewhere to determine the proper elevation, to select as large a catchment area as possible, and to take into consideration absorption, evaporation, and the character of the ground and underlying bed rock. The angle of slope for a peat dam, as given by J. T. Fanning,[b] is 2.75 horizontal to 1 vertical. Experience has shown that in Alaska, owing to the permanent frost, such dams will stand at a smaller angle, especially if brush is laid alternately with sod. The cost of storage dams in the interior may be reckoned at $1.75 per cubic yard of earthwork, and in Seward Peninsula at $1 per cubic yard.

The expedient of building settling ponds must be resorted to in places where a small amount of sluice water is used over and over for successive operations, as on Anvil Creek, in Seward Peninsula. Two such dams for retaining sediment, one of which is shown in Pl. XXII, *A*, are in use on Anvil Creek. The water is drawn off from these as often as possible. In the second case it is drawn off at intervals of ten hours to the supply flume.

## PIPE LINES.

In hydraulic mining the water is distributed from the pressure box to the monitors and elevators by means of wrought-iron or, more generally, steel-riveted pipe, usually made up in sections 17 to 19 feet in length. Sheet steel is used, from 8 to 16 U. S. standard gauge, bent, each plate, 30 or 36 inches in length, being riveted in double rows lengthwise and single on the ends. The sizes used in Alaska vary from 8 to 36 inches. The pipe is shipped by the manufacturers either made up and riveted, as above, ready to be laid with slip joints, or the material is supplied in short plate sections, bent, punched, and furnished with necessary rivets, baled and nested for transportation, ready to be cold riveted on the ground. Fig. 22 shows a form of ship-

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[a] Eng. and Min. Jour., Jan. 5, 1905.

[b] Treatise on Water Supply and Engineering, p. 315.

*A.* SETTLING POND, ANVIL CREEK, SEWARD PENINSULA.

*B.* PIPE LINE, ATLIN CREEK, BRITISH COLUMBIA.

ping made-up pipe of different diameters which renders it secure from bending or injury in transport. The iron bolt can afterwards be used in the blacksmith shop.

Pipe made up beforehand is coated by immersing it in a bath of asphaltum preparation. This is highly important, and if the operator rivets his pipe on the ground he should dip the pipe in such a bath before laying. Bowie gives the following formula for a bath:

| | Per cent. |
|---|---|
| Crude asphaltum | 28 |
| Coal tar (free from oily substances) | 72 |

The advantages of using sheet steel, commonly called "hydraulic" pipe, are its cheapness of construction, adaptability for crossing depressions (as in inverted siphons), the facility with which its position can be changed, and, when slip joints are used, the ease with which the line can be laid. Many engineers do not recommend the use of slip joints—that is, the slipping of the end of one length of pipe 3 inches into the end of the next to form the joint. With average topography, however, experience has proved that such joints stand great pressure, and the practice is nearly universal among hydraulic miners.

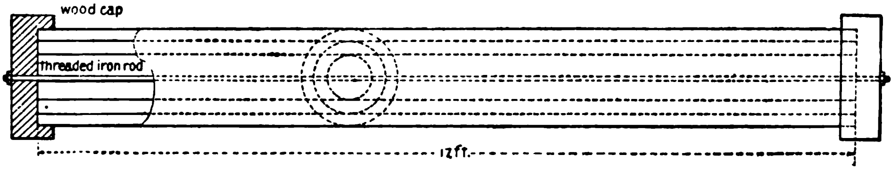

FIG. 22.—Method of nesting made-up hydraulic pipe for shipment.

Pipe is used for the three following purposes:

(1) As a water conduit, replacing ditches and flumes. Pipe is not recommended for this purpose except where siphoning is necessary or where conditions are particularly difficult for ditching and fluming.

(2) For leading water from the pressure box to the claim.

(3) For leading the water from the gate of a Y to its various points of discharge, as to giants, elevators, and impulse or "hurdy-gurdy" wheels.

The discharge pipe is generally a nozzle, either manipulated by means of a swivel-jointed tapering pipe called a giant or monitor or fixed within a larger pipe (the contrivance known as an elevator), or acting on the buckets of a wheel.

The thickness of the iron or steel employed is determined by the pressure of the water and the diameter of the pipe. Table 9 gives information concerning a few of the sizes used in Alaska. Prices are those prevailing in San Francisco.

TABLE 9.—*Double-riveted sheet steel slip-joint hydraulic pipe.*

| Diameter in inches. | Thickness, U. S. gage. | Thickness in fractions of an inch. | Head, in feet, pipe will safely stand. | Pressure in pounds per square inch due to head. | Weight per foot in pounds. | Price per foot, San Francisco, 1904. |
|---|---|---|---|---|---|---|
| 6 | 18 | $\frac{1}{20}$ | 480 | 208 | 3.80 | $0.23 |
| 6 | 16 | $\frac{1}{16}$ | 600 | 260 | 4.85 | .25 |
| 8 | 16 | $\frac{1}{16}$ | 450 | 195 | 6.30 | .32 |
| 8 | 14 | $\frac{5}{64}$ | 563 | 244 | 7.75 | .35 |
| 11 | 16 | $\frac{1}{16}$ | 325 | 141 | 8.50 | .37½ |
| 11 | 14 | $\frac{5}{64}$ | 408 | 177 | 10.50 | .44 |
| 11 | 12 | $\frac{7}{64}$ | 572 | 248 | 14.50 | .60 |
| 15 | 18 | $\frac{1}{20}$ | 191 | 83 | 8.50 | .46 |
| 15 | 16 | $\frac{1}{16}$ | 240 | 104 | 10.75 | .50 |
| 15 | 14 | $\frac{5}{64}$ | 300 | 130 | 13.00 | .55 |
| 15 | 12 | $\frac{7}{64}$ | 420 | 182 | 19.00 | .85 |
| 18 | 10 | $\frac{9}{64}$ | 450 | 195 | 30.00 | 1.50 |
| 22 | 16 | $\frac{1}{16}$ | 164 | 71 | 15.75 | .70 |
| 22 | 12 | $\frac{7}{64}$ | 286 | 124 | 28.50 | 1.20 |
| 24 | 16 | $\frac{1}{16}$ | 150 | 65 | 18.00 | .80 |
| 24 | 14 | $\frac{5}{64}$ | 187 | 81 | 22.00 | 1.00 |
| 30 | 16 | $\frac{1}{16}$ | 120 | 52 | 21.00 | .90 |
| 30 | 14 | $\frac{5}{64}$ | 150 | 65 | 27.00 | 1.10 |

Pipe-reducing sections, Y's, forks, and elbows, and the method of "nesting" pipe for shipment are illustrated by fig. 23 (p. 131). Sections of pipe are put together, as above stated, by slipping, or by flange or lead joints. If it is advisable to reenforce a slip joint, the simple device shown in fig. 24 (p. 132), which can be made quickly in the blacksmith shop, will be found useful. The sleeve, lugs, and key should be made of soft steel.

The disadvantage of diverting water from a straight pipe line may be illustrated by the experience of one of the Alaska operators. It was found that diverting the water from an 18-inch plugged pipe by means of 6-inch Y branch gave an efficiency of 2, while in using the same water through the direct 18-inch pipe choked to 6 inches the efficiency was 3.

In laying pipe from the pressure box to the claim the line should be started at the lower end and the joints slipped in down the slope. Various methods of "setting" the pipe are in use. The device shown in fig. 25 is used by Mr. F. H. Brackett, of Atlin, British Columbia. It consists of a square block of timber 3 by 3 feet by 9 inches, faced with one-sixteenth inch steel plate, to which is bolted a disk-like wooden plug the diameter of the pipe inside. Two men batter the timber with the mallet. Another device used in Oregon for assistance in setting and unsetting pipe is shown in fig. 26. The wrench is made with reversible parts, so that the position of the leverage can be changed for the different operations. Either of these devices can be easily made on the ground.

In cold climates it has been found good practice to lay the pipe line in a slight lateral curve down a slope, so that subsequent contraction of the units may be remedied by pushing the pipe into a more nearly straight line.

In laying the pipe line a funnel-shaped section of pipe for the water to run into is usually provided at the pressure box. This is of light gage and is 6 feet long and from 40 to 30 inches at its larger end. From this the pipe is generally of uniform diameter as far as the Y or iron gate, from which the water is distributed to various parts of the operations. The pipe should be laid as nearly straight as conditions will allow, and elbows and bends of small radius should be avoided.

Lead joints are seldom necessary in Alaska operations, but where there are sharp declivities pipe joints must be braced and strengthened by means of lugs and wiring, as illustrated in fig. 27. It is seldom possible to lay pipe on the "hydraulic grade line."[a] This is "an imaginary straight line, extending from a point on the side of the water box or reservoir, denominated the velocity head, to the mouth of the nozzle." When the pipe line departs greatly from the hydraulic grade line, allowances based on formulas given in the various textbooks on hydraulic mining must be made for the difference in pressure.

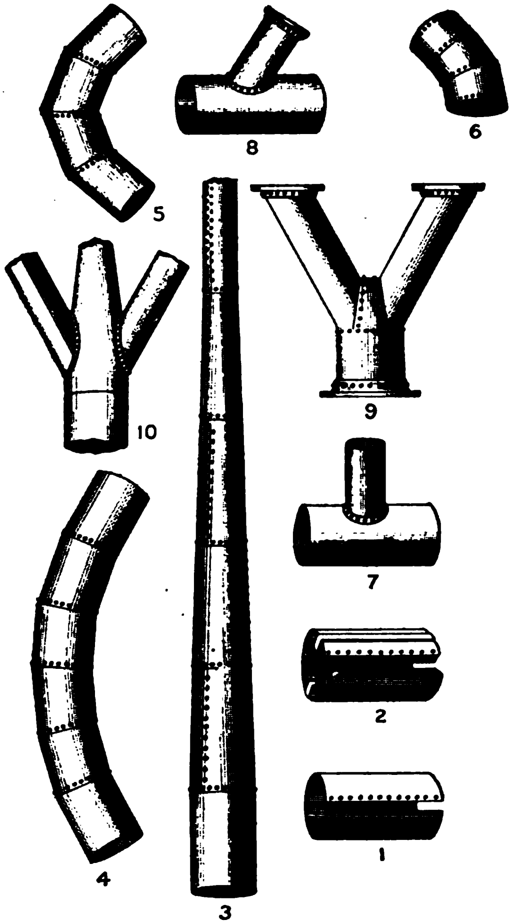

Fig. 23.—Forms of pipe fittings. 1, Pipe cut, formed, and punched, to be rivited on ground; 2, pipe nested for shipment; 3, reducing length; 4, circular bend; 5, angular bend; 6, short angular bend; 7, T discharge; 8, angular T discharge; 9, two-way Y; 10, three-way Y.

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[a] Van Wagenen, op. cit., p. 69.

Where 22- to 30-inch pipes are used it is not advisable to employ less than 14 gage, even under low heads, as lighter pipe will not bear handling. The practice of burying pipe, or at least sodding it over, where the line is to be in place for more than a season, is recommended in Alaska even more than in other countries.

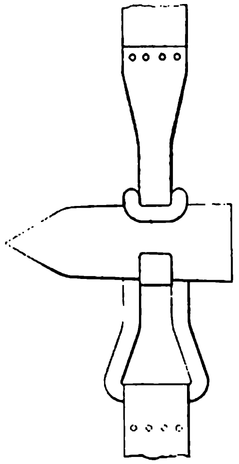


FIG. 24.—Device for reenforcing joints of pipe.

Air valves, which can be simply made of leather, hinged to the inside of 3-inch openings in the pipe, should be not more than 300 feet apart. These allow the air to escape when the pipe is being filled and prevent the collapse of the pipe when suddenly emptied of water. Leakage in joints may frequently be stopped by slowly feeding sawdust or manure in at the upper end. In case of collapse of a pipe line, experience has shown that by turning on the water slowly, after closing the lower end, the collapsed sections may be restored to shape for temporary use. The experiment is risky.

Pl. XXII, *B*, shows the set-up of a hydraulic pipe line on McKee Creek, Atlin. As may be seen, the line is well braced and rock ballasted, and on steep places the joints are held by lugs and wire guys. The line, which is one of three in use on this property, is of 12-gage steel, is 1,200 feet in length, has a 30-inch mouthpiece at the penstock, and is successively reduced to 18, 16, and 12 inches, the last discharging through the giant. At full capacity the discharge was said to be 700 miner's inches at 170-foot head. The discharge was through a No. 4 Vancouver giant, using 6-inch nozzle, the average twenty-four hour duty being 4 cubic yards of gravel. Along the pipe line side ditches were cut for a short distance to take care of leakage. Pl. XXIII, *A*, shows a pipe line led down a hillside on Pine Creek, Atlin.

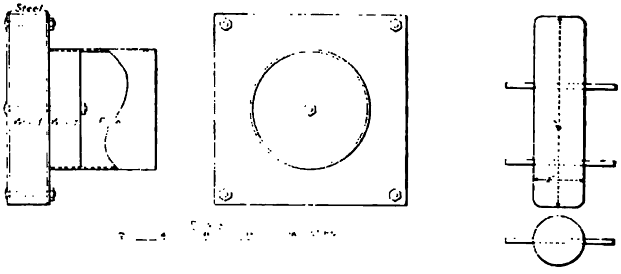
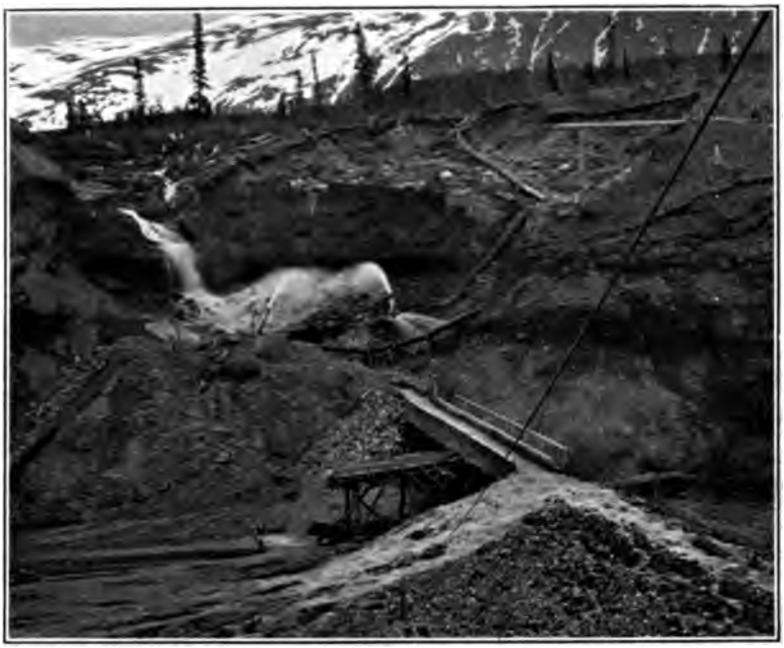


FIG. 25.—Device for setting pipe by battering.

An important fact regarding the flow of water in pipes is the loss of effective pressure at the nozzle due to friction, owing to the use of *pipes* of too small diameter. The friction of water in pipes increases



A. SET-UP OF HYDRAULIC PIPE LINE, McKEE CREEK, ATLIN, BRITISH COLUMBIA.

B. HYDRAULIC PIT, KLONDIKE BENCHES.

as the square of the velocity and also depends on the condition of the pipes. Even the rivet heads in a pipe line cause friction and consequent loss of head. As Mr. George H. Evans[a] points out, it makes no difference whether the water is flowing up hill or down, or the pressure great or small, the total friction will be materially the same.

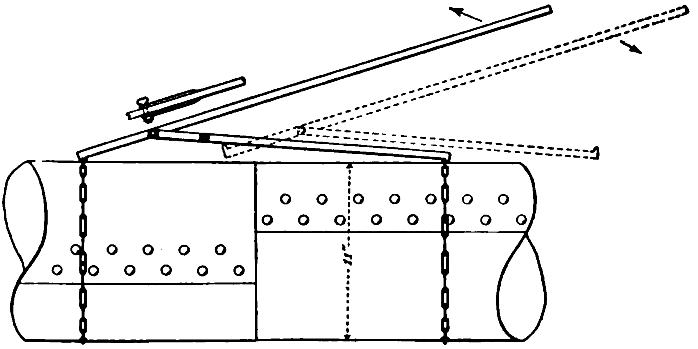


FIG. 26.—Pipe wrench for setting and unsetting hydraulic pipe.

In wooden pipes the friction is nearly double that in iron or steel pipes. Cox's formula for finding the friction head, which must be subtracted from the actual head to give the effective head, is as follows:

$$H=\frac{L}{1200d}\times(4\times V^2+5V-2)$$

$H$ =friction head in feet.
$d$=diameter of pipe in inches.
$L$ =length of pipe in feet.
$V$=velocity of water in feet per second.

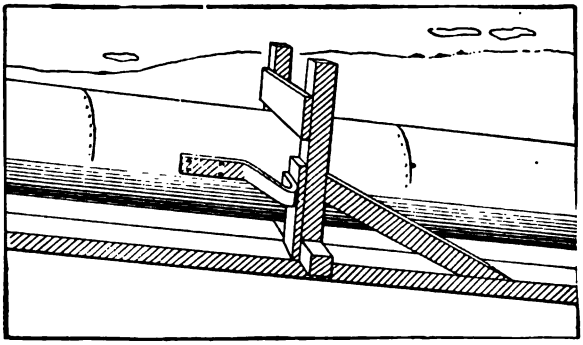


FIG 27.—Method of bracing hydraulic pipe on steep slope.

The loss in head of a pipe line 12 inches in diameter, discharging 400 miner's inches, and 5,000 feet in length, is computed by Mr. Evans to be 246.44 feet. Or, if the actual head is 500 feet, the effective head is reduced to 253.56 feet, or an original pressure of 217 pounds per square inch to 110 pounds. To find the velocity in feet per minute in a pipe, multiply the number of cubic feet of water discharged per minute by 144, and divide the product by the area of the pipe in square inches. The velocity in feet per second can then be found.

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[a] Practical Notes on Hydraulic Mining, San Francisco, 1898, p. 25.

Experiments by Mr. Hamilton Smith[a] appear to demonstrate that with new and carefully coated pipes, with velocity as high as 11 feet per second, the loss of head is not significant. Nevertheless, the numerous conditions which enter into the laying of pipe lines on placer mines render the rule that the velocity should not exceed 3 feet per second in the pipe a very safe one to follow. It is a common fallacy among men inexperienced in hydraulicking that an increase in head will increase the amount of gravel which can be moved to the sluice. It must be remembered that the force of the water imparted by head is entirely expended in piping against the face, while the sluice is the governing factor in the moving of gravel after it leaves the face. The aid given by the water in moving gravel to the sluice, and in it is dependent on the grade over which it runs. In small hydraulic operations water is conducted to the pit from the head ditch or penstock by means of canvas hose, known as flume hose, which is from 6 to 14 inches in diameter and weighs from 8 to 15 ounces per foot. This is used mainly on account of its cheapness and, as a less important consideration, because of the ease with which it may be moved about. It is used also for conveying water with very low head, for sluicing purposes.

Flume hose is at best a makeshift and its use is not recommended, except in operations of primitive character in very remote districts. It should never be used to handle more than 100 inches of water or for a head greater than 50 feet.

## GIANTS AND NOZZLES.

### SIZES AND PRICES.

The practice in using hydraulic discharge pipes, known as monitors, giants, and nozzles, does not differ in Alaska from that pursued elsewhere.

Table 10 gives the principal facts necessary for the miner to know about hydraulic giants of the size most applicable for Alaskan work.

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[a] Smith, Hamilton, Hydraulics, 1886, p. 314.

TABLE 10.—*Water required, effective work, sizes, volumes, and heads of water, weights and prices of double-jointed hydraulic giants.*

[Prices include two nozzles to each machine and are subject to discount.]

| No. of giants. | Diameter of pipe inlets. | Diameter of butts with nozzle detached. | Size of nozzles. | Effective head or pressure. | Volumes of water necessary for effective work. | | Weights of heaviest part. | Shipping weights. | List price, 1904. | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | Double-jointed, ball-bearing. | Double-jointed, plain. |
| | *In.* | *In.* | *In.* | *Feet.* | *Miner's inches.* | *Miner's inches.* | *Lbs.* | *Lbs.* | | |
| | | | | | 1-inch nozzle. | 1½-inch nozzle. | | | | |
| 0 | 5 | 2½ | 1, 1½ | 100 | 20 | 45 | 100 | 330 | $100 | $85 |
| | | | | 150 | 25 | 55 | | | | |
| | | | | 200 | 30 | 65 | | | | |
| | | | | | 2-inch nozzle. | 3-inch nozzle. | | | | |
| 1 | 7 | 4 | 2, 3 | 100 | 80 | 185 | 120 | 390 | 140 | 125 |
| | | | | 200 | 115 | 260 | | | | |
| | | | | 300 | 140 | 320 | | | | |
| | | | | 400 | 160 | 365 | | | | |
| | | | | | 3-inch nozzle. | 4-inch nozzle. | | | | |
| 2 | 9 | 5 | 3, 4 | 100 | 185 | 325 | 150 | 520 | 195 | 175 |
| | | | | 200 | 260 | 460 | | | | |
| | | | | 300 | 320 | 565 | | | | |
| | | | | 400 | 365 | 650 | | | | |
| | | | | | 3-inch nozzle. | 4-inch nozzle. | | | | |
| 3 | 11 | 6 | 3, 4 | 100 | 185 | 325 | 210 | 890 | 250 | 225 |
| | | | | 200 | 260 | 460 | | | | |
| | | | | 300 | 320 | 565 | | | | |
| | | | | 400 | 365 | 650 | | | | |
| | | | | | 4-inch nozzle. | 6-inch nozzle. | | | | |
| 4 | 11 | 7 | 4, 6 | 100 | 325 | 730 | 225 | 1,075 | 275 | 250 |
| | | | | 200 | 460 | 1,000 | | | | |
| | | | | 300 | 565 | 1,270 | | | | |
| | | | | 400 | 650 | 1,460 | | | | |

About 70 giants of several different makes were in operation in the territory visited during the summer, of which 58 per cent were of size No. 2, using commonly a 3-inch nozzle. In the Juneau and Atlin districts size No. 4, using 4- and 6-inch nozzle, is used as a rule, while in the interior and Seward Peninsula the sizes 0, 1, and 2, using from 2- to 4-inch nozzle, are preferable. The double-jointed, ball-bearing giant is the favorite type. Deflectors are not commonly used with sizes below No. 2, while in working on the Klondike benches the practice is in favor of butt ends even with larger giants.

The giants used in Alaska are almost exclusively of California manufacture, although in the Atlin and Klondike districts some of Canadian make are used. The importance of using water as free as possible from solid matter in suspension should be considered and great care should be taken to trap all foreign material at the pressure box. The rifling in the barrel of the monitor and the boring of the nozzle are also of the first importance, and operators will find it expensive to use other than standard and well-tried makes of giants. A method of bracing giant is shown in fig. 28. In soft schist bed rock, however, it is frequently necessary to brace with deadman and cable.

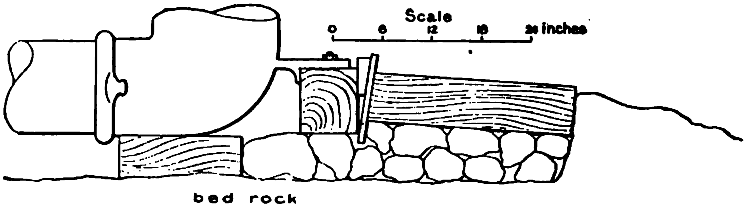


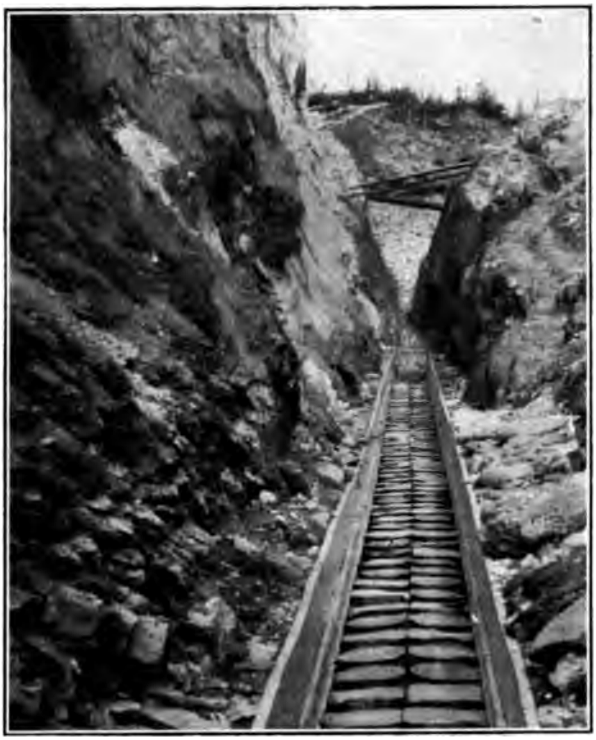
FIG. 28.—Method of bracing giant.

### METHODS OF PIPING TO SLUICE.

It has been found that some operators in the north are in favor of setting the giant on the top of the bank and piping downstream. This is the practice on McKee Creek, Atlin district, and in the Nome and Council districts. Although the duty attained by the miner's inch does not prove this to be advantageous, it should be remembered that the height of bank to be operated on is generally small—from 15 to 50 feet. There is little caving to be done with the pipe, and except in the case of the rounded gravels of the Klondike "White Channel," it is thought that the water under pressure assists in moving the material to the sluice. Experience does not indicate that the attempt to drive the gravel with the nozzle is very effective.

The various precautions concerning keeping the gravel bank square, the prevention of dangerous caves, and the necessity of continuous work during the short season are well known to the hydraulic miner in any country. In working the shallow gravel banks of Alaska it should be borne in mind that the giants must be frequently moved and the tail sluice extended. These operations consume time, and the most expeditious system possible for performing them should be adopted at the commencement of the short season.

Fig. 29 shows the method of rigging up a hydraulic mine on one of the narrow benches of White Channel gravel bordering Bonanza Creek in the Klondike, and Pl. XXIII, *B*, shows a portion of the ground which has been worked as indicated in the sketch.



*A.* BED-ROCK SLUICE, KLONDIKE BENCHES.

*B.* BLOCKS OF FROZEN GRAVEL IN HYDRAULIC PIT, KLONDIKE.

In piping against a 40-foot bank of frozen gravel, as here illustrated, it has been found good practice to use the giant against a certain part of the bank from two to six hours a day, and allowing the gravel to thaw the remainder of the twenty-four hours. Piping continuously against a frozen bank is a waste of water and power. (See Pl. XXIV, *B*.) The use of powder is of no avail. Even if the gravel is broken off and moved to the sluice in frozen chunks it can not be washed nor can the gold be extracted from it. A combination of water under pressure and of the action of the sun, rightly adjusted, is most effective in thawing frozen gravel. Mr. J. P. Hutchins says that for 200 to 250 miner's inches of water a face covering 50,000 square feet should be allowed on the shady side of hills, while a much less surface will suffice on a sunny slope.

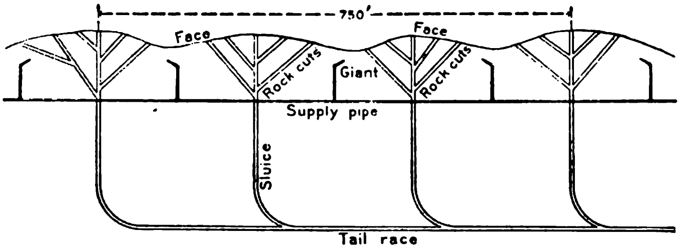


FIG. 29.—Method of hydraulicking frozen gravel, Klondike benches.

It will be noticed that in the Klondike operations here figured deep trenches or "ground sluices" are cut in the bed rock leading from the face to the head of the tail sluice. The cutting and constant extension of such trenches is a necessity in almost any hydraulic operation, to a greater or less extent, and forms a considerable item of expense. At the time here figured four men in twenty-four hours were employed all the time in blasting out the bed-rock cuts, the expense amounting to not less than $50 a day. In some cases the bench miners have sunk a shaft at a central part of the ground and run a long tunnel on the grade of the sluice, to which all the gravel is moved. Although such tunnels carrying the tail sluice will be from 200 to 500 feet in length, and will cost from $3,000 to $10,000, they will generally be found less expensive than the constant cutting of trenches to connect with the main sluice carried as shown in the plant.

## BANK-HEAD WATER.

In no case seen in the North has the amount of bank-head water been excessive. This is a useful accessory to the mining operations, and where it is available should be used. The width of tail sluices should be great enough to allow for excess water, as at times of local rains such water can frequently be obtained and a larger amount of gravel thereby assisted in reaching the sluice.

Pl. XV, *B* (p. 92), shows hydraulic operations in Silver Bow basin, Alaska, and illustrates the use of the bank-head water.

### DUTY OF A MINER'S INCH.

This term is applied to the quantity of material moved by 1 inch of water in twenty-four hours. It depends on quantity of water, character of material washed, height of bank, size and grade of sluice, and kind of riffle. In many mines the gravel may be easily broken down and carried to the sluice, but may be very hard to move through the sluice on account of a light grade, disproportionate width of box, or the use of obstructive riffles. Thus, according to Bowie, in the North Bloomfield mine the duty varied from 3.86 to 4.8 cubic yards, with 100 to 265 feet of bank, sluice 6 feet wide by 32 inches deep, grade 6½ inches in 12 feet. At La Grange mines, on the other hand, the duty was from 1.08 to 1.82 cubic yards, height of bank 50 to 80 feet, sluice 4 feet wide, 30 inches deep, and grade 3 inches in 12 feet. In both cases the riffle pavement was principally blocks.

Table 11 (p. 139) is instructive as showing the variation in duty of the miner's inch under the different governing conditions in the North. The duty of the miner's inch in the Klondike is large, estimated at 8 cubic yards in twenty-four hours in the operation described on page 137, with water under 130-foot head and a grade of 12 inches to 12 feet in the sluice boxes, a variable amount of bank-head water from 25 to 100 miner's inches being used. The high duty is accounted for by the fact that the material washed is well rounded, by the absence of large stones, heavy grades to sluices, and the fact that block riffles are generally employed.

The low duties at Nome are accounted for principally by the fact that one-half to two-thirds of the water is generally diverted for use in the hydraulic tailings lifts, and partly by the fact that the gravel is flat and rough. Iron riffles are generally used, but this factor plays little part on account of the short sluices in use.

TABLE 11.—*Duty of the miner's inch in northern placer mining.*

[The estimates are based on statements made by the operators.]

| Locality. | Height of bank. | Grade. | Miner's inches of water. | Cubic yards duty. | Sluice dimensions. | | Riffles. | Remarks. |
|---|---|---|---|---|---|---|---|---|
| | | | | | Width. | Depth. | | |
| | *Feet.* | | | | *Inches.* | *Inches.* | | |
| Juneau: | | | | | | | | |
| Windfall Creek.... | 16 | ........................ | 1,000 | 2 | ........ | ........ | .............................. | |
| Gold Creek........ | 200 | 4½ inches : 12 feet..... | 4,000 | 2 | 72 | 60 | Blocks..................... | Heavy stones. |
| Silver Bow basin.. | 80 | 4 inches : 12 feet...... | 2,500 | 2 | 50 | 55 | .....do..................... | Do. |
| Atlin: | | | | | | | | |
| McKee Creek...... | 40 | 8 inches : 12 feet...... | 700 | 1½ | 32 | 32 | Angle iron and blocks...... | Do. |
| Do............ | 85 | 8 inches : 12 feet...... | 1,200 | 3 | 32 | 40 | .....do..................... | Do. |
| Birch Creek....... | 25 | 5¼ inches : 12 feet..... | 1,200 | ¼ | 30 | 30 | Blocks..................... | Do. |
| Spruce Creek...... | 29 | 5 inches : 12 feet...... | 1,200 | 1 | 40 | 36 | .....do..................... | Do. |
| Do............ | 20 | 5½ inches : 12 feet..... | 900 | ¼ | 48 | 32 | .....do..................... | Do. |
| Pine Creek........ | 20 | 3 inches : 12 feet...... | 700 | ⅔ | 36 | 36 | Long rails.................. | Do. |
| Do............ | 60 | 5 inches : 12 feet...... | 3,500 | ½ | 60 | 40 | Block....................... | Do. |
| Dawson: | | | | | | | | |
| Bonanza Creek.... | 20 | 12 inches : 12 feet..... | 250 | 5 | 24 | 20 | Blocks..................... | "White Channel" small round gravel, frozen. |
| Do............ | 25 | 12 inches : 12 feet..... | 125 | (?)4 | 20 | 16 | Sawed pole, iron-shod...... | Hillside, small gravel, frozen. |
| Do............ | 25 | 11 inches : 12 feet..... | 150 | 10 | 24 | 24 | Blocks..................... | "White Channel" small round gravel, part tailings. |
| Do............ | 35 | 12 inches : 12 feet..... | 200 | 6½ | 24 | 24 | .....do..................... | "White Channel" small gravel, frozen. |
| Do............ | 75 | 12 inches : 12 feet..... | 266 | 8 | 24 | 30 | .....do..................... | Do. |
| Eldorado Creek.... | 60 | 11 inches : 12 feet..... | 160 | 7 | (?)24 | 24 | Iron, Hungarian, and blocks | Do. |
| Last Chance....... | 46 | 12 inches : 12 feet..... | 230 | 7 | 24 | 18 | Pole, iron-shod, and blocks. | Do. |
| Do............ | 25 | 14 inches : 12 feet..... | 120 | 5 | 20 | 20 | Pole, iron-shod............. | Do. |
| Hunker Creek..... | 15 | 13 inches : 12 feet..... | 150 | 2 | 14 | 14 | Pole and blocks............. | "White Channel," very little frozen. |
| Do............ | 8 | 12 inches : 12 feet..... | 125 | 2¾ | 17 | 14 | Pole, iron-shod............. | "White Channel" small gravel, frozen. |
| Nome: | | | | | | | | |
| Anvil Creek....... | 30 | 4½ inches : 12 feet..... | 500 | 1¾ | 33 | 18 | Angle iron.................. | Heavy partly frozen gravel, much flat schist. |
| Do............ | 20 | 4½ inches : 12 feet..... | 400 | 10 | ........ | ........ | .....do..................... | Frozen earth or "muck" stripping. |
| Do............ | 12 | 9 inches : 12 feet...... | 100 | ........ | 16 | 14 | .....do..................... | |

TABLE 11.—*Duty of the miner's inch in northern placer mining*—Continued.

| Locality. | Height of bank. | Grade. | Miner's inches of water. | Cubic yards duty. | Sluice dimensions. | | Riffles. | Remarks. |
|---|---|---|---|---|---|---|---|---|
| | | | | | Width. | Depth. | | |
| Nome—Continued. | *Feet.* | | | | *Inches.* | *Inches.* | | |
| Anvil Creek | 4½ | 3 inches : 12 feet | 100 | 3 | (a) | (a) | | Frozen muck, 75 per cent ice, ditch head. |
| Glacier Creek | 20 | 6 inches : 12 feet | 760 | 1.32 | 36 | 24 | Angle-iron grates | Two-thirds water used for lifting 36 feet flat semi-frozen gravel. |
| Dexter Creek | 40 | 6½ inches : 12 feet | 100 | (?)2 | 16 | 16 | ....do | Heavy gravel, limestone bed rock. |
| Newton Creek | 8 | 7 inches : 12 feet | 150 | 3½ | 24 | 24 | Pole and angle iron | Small coastal plain gravel. |
| Basin Creek | 20 | 8 inches : 12 feet | 250 | .6 | 18 | 18 | Pole, iron-shod | Elevator using three-fifths water, 29 feet lift, heavy gravel. |
| Council: | | | | | | | | |
| Ophir Creek | 12 | 10 inches : 12 feet | 600 | 1½ | 24 | 16 | Rails and angle iron | Elevator using three-fifths water, 29 feet lift, heavy gravel. |
| Do | 8 | 10 inches : 12 feet | 750 | 1 | 24 | 16 | ....do | Elevator using two-thirds water, 28 feet lift, slabs and clay. |
| Crooked Creek[b] | 4 | 4½ inches : 12 feet | 55 | 3.45 | (a) | (a) | | Frozen muck, ground sluice, ditch head. |
| Solomon: | | | | | | | | |
| Solomon River | 5 | 10 inches : 12 feet | 600 | .8 | 37 | 24 | Angle-iron grates | Elevator using one-half water lift, 12 feet subangular gravel. |

[a] Ground sluice.

[b] Ground sluicing of frozen "muck" material consisting of 50 to 75 per cent ice.

The duty given above is based entirely on statements made by the operators, and owing to the short amount of running time which has furnished the data, no particular case can be considered of high authority. It will be seen, however, that the different operations in each separate province present a certain amount of agreement.

In stripping operations where the "frozen muck" is removed by water, ground sluicing appears to be nearly as efficient as the use of water through the nozzle, both in the interior and Seward Peninsula. This peculiar material, from 50 to 75 per cent ice, is easily thawed when exposed to the atmosphere and to flowing water. It appears rather to be melted than cut away, therefore the impact of a hydraulic stream adds little to the efficiency of a given quantity of water.

It is stated by the Klondike operators that were the "White Channel" gravels unfrozen the duty of the inch would be twice as large. This is not impossible, as the bench gravel presents most favorable characteristics for easy handling. In its frozen state, however, the bank has the consistency of fresh granite.

In Seward Peninsula the instances tabulated are mostly those where hydraulic lifts are employed. The duty has been given in terms of the total amount of water used, including bank head, monitor, and elevator water. In the cases where elevators are used the grade and dimensions refer to the bed-rock sluice leading to the throat, and do not refer to the tail sluice.

## HANDLING OF BOWLDERS.

The handling of large bowlders in northern hydraulic operations does not present serious problems. Where hydraulicking is possible it is generally found that all the material is small enough to go through the sluice. In the districts visited during the season's work the Atlin district, of northern British Columbia, was characterized by gravels containing large bowlders, which had to be derricked out or sledged in the pit. The practice of "bulldozing" or "plastering" bowlders to break them, by placing sticks of powder on top, covered with a mud cap, is not to be recommended on account of the expense of powder. Sledging into pieces small enough to be sent to the sluice will generally be found the cheapest method, especially as bowlders consisting of hard and tough rocks are not frequently encountered.

Mr. J. D. Hauer, in a recent article,[a] states that in general contract work under given conditions the cost of sledging sandstone bowlders varied from 1 to 9 cents per cubic yard; "mud capping" or "bulldozing" costs from 18 cents to as much as 51 cents; "blocking," or breaking by drilling and blasting, costs from 11 to 18 cents; while the method of placing the powder charge under the bowlders costs from

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[a] Engineering News, quoted in Mining and Scientific Press, Feb. 9, 1905.

15 to 17½ cents per cubic yard. He recommends the method last mentioned as against bulldozing, or placing charge on top, as powder is saved and there is the additional advantage that much of the rock is thrown out of the cut.

The practice of installing large and expensive derricks as an adjunct to hydraulic mining is not generally recommended for Alaskan work.

### SLUICES AND GOLD-SAVING APPLIANCES IN HYDRAULIC OPERATIONS.

In the South Coast region the plant of the American Gold Mining Company in Silver Bow basin is characteristic. This sluice, while not in use during the season of 1904, has been operated since 1901, and is typical of the most favorable conditions under which hydraulic tail sluices can be constructed in southeast Alaska.

A 9- by 10-foot tunnel, 3,300 feet in length, was driven through a spur of the mountain from the bank of Gold Creek to tap the gravel pit at the proper depth for reaching the lowest sag in the bed rock. The grade of the tunnel, like that of its contained sluice, is 4 inches to 12 feet. It was driven with air at the rate of 8 feet a day in slate, at a cost of $20 per foot. No timbering is used, with the exception of a few sets near the ends.

The sluice running through the tunnel is 4 feet wide and 4 feet 10 inches deep, inside measure, of 2-inch native lumber, lined with 1-inch "sand" or lining boards, all lumber planed and sized. The sills are laid on bed rock, and are 6 inches square and 5 feet long, the posts being also of the same size. Posts and sills are braced by 1 by 8 inch pieces. The sluice is 3,700 feet long from the tunnel entrance and extends for 900 feet up the bed of the pit through the worked-out ground. Bowlders as large as 10 inches in diameter are handled in this sluice without difficulty. The amount of water used varies according to the season, but averages 2,500 inches. Riffles are 12- by 12- by 12-inch spruce blocks set on end, separated by 1½- by 2-inch strips, set on edge and nailed with headless nails. The duty of a miner's inch of water is 2 cubic yards per twenty-four hours. The amount of lumber necessary for constructing one box of a sluice, such as the above, approximately 1,100 feet, is probably excessive for the needs of the case. Local conditions in the North frequently result in increased expenditure for the sake of hastening work. About 25 pounds of nails were consumed to each box, and $10 worth of labor. The cost will not fall greatly below $30 per box of 12 feet, exclusive of cost of tunneling and shooting out the bed rock to grade in carrying up the boxes through the pit. The annual cost of maintenance, including renewal of riffle blocks, which last two years, and renewal of lining boards, is approximately $1,000 a year.

In the saving of the gold, which is fine and rough, 23 flasks of quicksilver are distributed in the boxes. One flask of quicksilver is consumed per month.

Six undercurrents are placed near the end of the sluice, within a length of from 300 to 500 feet of the end. They are transversely arranged, 24 by 14 feet in dimensions, and on a grade of 10 inches to 12 feet. They are furnished with Hungarian riffles—wooden strips with flat strap iron nailed to the top.

The first 600 feet of the sluice are cleaned up monthly, and in this length 75 per cent of the gold is said to be saved. The remainder of the sluice is cleaned up once a year. The experience with the undercurrents was not satisfactory, as very little gold was caught in them.

It should be especially noted that in the South Coast province the placer gold is of recent origin, is not removed more than a mile from its original source, is low in fineness (in the present case $16.50), and is bright and rough. However finely divided, such gold is easy to save, and the advantage of undercurrents may be questioned. In the interior of Alaska the case is different, as will be seen.

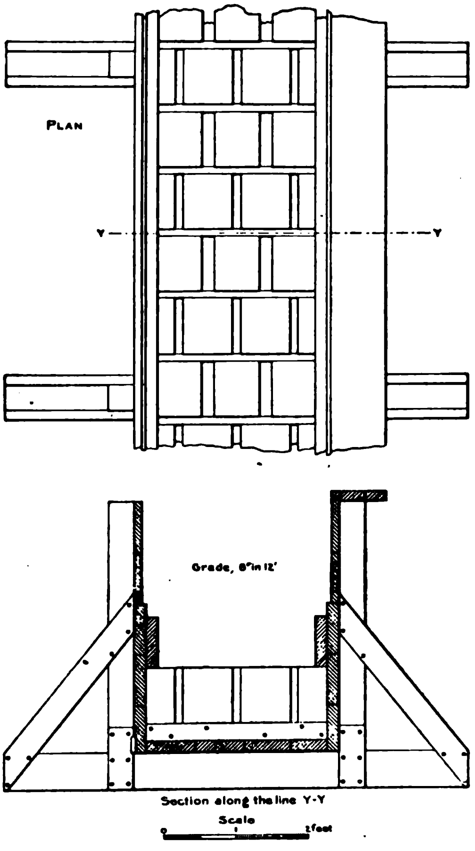

FIG. 30.—Sluice box used on McKee Creek, Atlin, British Columbia.

In the Atlin district of British Columbia the grades attainable in the tail sluices are generally low, from 3 to 5 inches to the box length of 12 feet. McKee Creek forms an exception to this rule, the grade being 8 inches in the sluices of both the plants now operating there. The practice of the Amalgamated McKee Creek Mining Company on this creek is as follows:

The sluices, of which there are two, are 600 and 700 feet long. The grade is 8 inches to 12 feet, and the inside dimensions of the sluice, exclusive of lining boards, are 29 inches wide by 38½ inches deep, as illustrated in fig. 30.

The amount of lumber in a sluice box like that figured is as follows:

8 posts, 4 by 4 inches by 3⅓ feet.
4 sills, 4 by 6 inches by 6 feet.
16 braces, 1½ by 4 inches by 1 foot.
1 top rail, 1½ by 8 inches by 12 feet.
16 post straps, 1¼ by 2 by 4 inches.
3 bottom boards, 1½ by 8 inches by 12 feet.
1 bottom board, 1½ by 4 inches by 12 feet.
4 side boards, 1 by 8 inches by 12 feet.
6 side boards, 1½ by 8 inches by 12 feet.
2 lining boards, 1½ by 8 inches by 12 feet.
48 riffle blocks, 8 by 8 by 12 inches.
16 riffle strips, 1 by 3 by 28 inches.

Each box, with riffle blocks, contains approximately 540 feet of lumber.

The use of 8-inch lumber is a necessity if native timber is used, 12-inch lumber being very scarce and the expense of employing it prohibitive. The cost of each box, including riffles, averages $25, sluice lumber being $45 per thousand and the riffle blocks costing $6 per box length. The setting of the riffle blocks with alternate spacing, as represented, is said to be advantageous, causing a cross circulation and consequent stirring action. The riffles are nailed in sets of three to a riffle strip outside the flume, and are piled up ready to be put in position as occasion demands. Pole riffles, 12 feet in length, made of 3- by 3-inch strips, set with a right-angle uppermost and shod with angle iron, are used in a portion of the boxes. A space is left between the lowest angle and the bottom of the box, the long strips being set with a gain into cross strips, 6 feet apart. The blocks are being gradually replaced by this type of pole riffle. The gold is coarse, and it is said that undercurrents would be of no advantage. In the first box 85 per cent of the gold is caught. In the first five boxes all the gold is caught which pays to clean up. In one clean-up $25,000 was caught in the first box, as against $1,900 in the remainder of the 700-foot sluice. The clean-up of the upper two boxes takes place once in two weeks. In this sluice the miner's inch is said to have a duty of 4 yards. A new box is put on the end of the tail sluice every two days. When the end of the sluice is spread into Y's, a hinged gate, with steel plates on both sides, is used for diverting the water and tailings into one or the other branches.

Experience on McKee Creek appears to show that the gold is saved in a comparatively short distance. Yet gravel miners in California maintain, and with apparent reason, that the longer the sluice the more gold will be saved. For example, at the Hidden Treasure drift mine, in Eldorado County, Cal., it was stated to the writer that the 2,500 feet of sluice boxes, 19 by 21 inches in the clear, with iron car wheel and rack riffles, and an undercurrent attached, were insufficient to save the gold. The sluice has a grade of 14 inches to 12 feet, and 90 pounds of quicksilver were used in charging the sluice.

The percentage of gold in the upper 120 feet of the sluice was as follows:

| | Per cent. |
|---|---|
| In upper 1 foot | 75.0 |
| In upper 10 feet | 8.4 |
| In next 50 feet | 9.6 |
| In next 60 feet | 2.0 |

This portion of the sluice was cleaned up monthly, and was furnished with old car wheels, regarded as the best riffle, and with iron Hungarian riffles.

The upper 1,000 feet of the sluice was cleaned up once in three months, and the whole sluice and undercurrents annually. Five per cent of the total product was obtained from the lower 2,380 feet paved with quartz stones, and it is estimated that an amount equal to 2 per cent of the total product escaped into the canyon below. It was stated that one-half mile of this canyon was leased to the Chinese for $5,000 per annum, who cleaned up at a profit the gold which had escaped. The undercurrent was not regarded as of great value, except for the saving of excessively fine gold and free quicksilver and gold with quartz attached.

Mr. W. M. Johnson informed the writer that in Nevada County, Cal., in the case of a sluice 1½ miles in length, the first five out of a total of seven undercurrents proved an economic success, the last two hardly paying to clean up. It was found best to clean the five undercurrents at intervals of two weeks.

The experience on Pine Creek, Atlin, in the matter of saving gold is not so satisfactory as on McKee Creek, according to operators' statements. Pl. XXIII, *A* (p. 132) shows one of the 120-foot sluices of the Pine Creek Power Company. It is 5 feet by 40 inches, has a grade of 5 inches in 12 feet, and is paved with block riffles. The undercurrent, 14 by 24 feet, on a 10-inch grade, was one of the few installations of the kind seen in the northwest. It was understood to be but the beginning of experiments aiming to save the fine gold that escaped with an excessive amount of black sand. The amount of gold recovered is 43 cents per cubic yard, as stated, and the opinion was expressed that a large percentage of fine gold escapes. The undercurrent itself had not been installed a sufficiently long time to determine its efficiency.

The Atlin gold is derived from older gravels through which the present streams cut. The gold is for the most part coarse, but is well rounded. That a certain percentage of it is fine, specimens collected show. It is very probable that an undue proportion of such bench gold will escape from a sluice of the short dimensions figured.

Drop-offs are not used in any of the sluices seen in Alaska or the North. Grade is lacking, it is true, yet in the case above mentioned there was a space of 25 feet vertically between the end of the sluice and the creek bed which served as the floor of the dump. One or two drops of 6 inches in the short sluice would seem entirely practicable, and would certainly be advisable. Cemented gravel does not occur generally at Atlin, but layers of clay are found in the pay streak. Such clay can not be properly disintegrated in a short sluice, but will roll in balls through the entire string of boxes, carrying with it not only the gold inclosed in its mass, but picking up gold already lying at rest.

A useful device to tighten the bottom boards of sluice boxes before nailing is shown in fig. 31. It is claimed that no tongue and groove are necessary with this appliance.

In the Atlin district the mining operations are hindered by heavy floods in the early part of June. At the property of the Société Minière on Bowlder Creek, it is the practice to cover the sluice with a series of small protective dams, so that the débris brought down by the annual flood may not injure the boxes. This débris is afterwards piped off, at the cessation of the high water, and the dams are then removed.

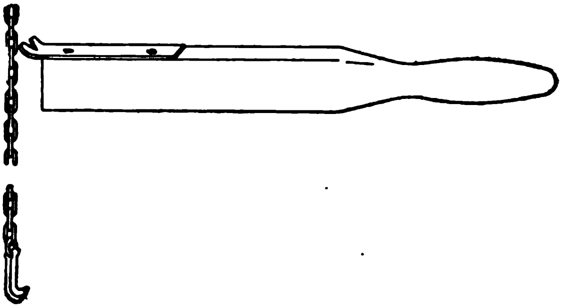

FIG. 31.—Side-board tightener used in making sluice boxes, Atlin, British Columbia.

The following details of this plant were kindly furnished by M. Henri Maluin, of the Société Minière de la Colombie Britannique, and will be of interest:

The property on Bowlder Creek operated by the Société Minière de la Colombie Britannique is situated in the Atlin mining division of the Cassiar district, British Columbia, 12 miles from the town of Atlin, on Atlin Lake. Operations were started in June, 1901, and the property has been worked continuously during the open season up to the present time. The product of gold has been 3,440 ounces, worth $16.63 per ounce. The gold is rather coarse, and nuggets up to 10 ounces in weight are found. Work is carried on both in the present bed of Bowlder Creek and on benches lying at approximately the same level as the creek bed. The gold-bearing channel is about 150 feet wide, the grade of the creek bed varying from 6 per cent to as high as 10 in the upper end. Where work is now carried on it is 6.1 per cent. The depth of ground averages from 25 to 50 feet, of which the lower 6 feet is considered the pay. The bed rock is a soft slate of greenstone variety, although granite occurs at the head of the creek. The gravel to be moved is rather heavy in character, stones from 18 to 30 inches in diameter composing 50 per cent of the mass, though the usual diameter of the stones approximates 6 inches.

In the season of 1904, a season being reckoned at 100 days, with 400 miner's inches of water available, 40,000 cubic yards of gravel were moved, at a cost not far from 50 cents per cubic yard. It is expected that mining will be done in the future for 40 cents per cubic yard, including all expenses. Mr. Maluin states that 35 cents would be a low estimate for the Atlin district, in general, for hydraulic work under the conditions obtaining on Bowlder Creek. Dirt is occasionally moved for 10 cents a cubic yard, and again patches of hard gravel are encountered which cost over a dollar.

An average of 25 men are employed during the summer season at wages of $3 per day of ten hours, and board. The season is open from June 1 to September 15. Spruce and pine are obtainable on the property, though it is difficult to obtain boards wider than 8 inches. Sawed flume lumber costs $40 per thousand.

The water for hydraulicking purposes is brought to the claim from higher up in Bowlder Creek by a flume 3,000 feet in length, size 24 by 24 inches, having a grade of 1 inch in 12 feet. It has an estimated capacity of about 700 miner's inches, but actually 400 inches is generally available at 150-foot head. The pipe line is 16-inch steel to the Y, then 14-inch to the two giants. Vancouver pipe is used, but the giants are California make. The average height of bank is 50 feet, and the giants have no difficulty in cleaning bed rock.

The sluice is 1,400 feet in length and cost $6,000. It is 24 inches wide on a grade of 4 per cent, corresponding very nearly to 6 inches to the 12 feet. Bottom boards are 1½ inches thick, sides 1¼ inches thick, while sills and posts are 4 by 6 inches. It is possible to keep a drop below the end box of 35 feet for dump. In piping, the head box is kept within 30 feet of the bank, and four men for one-half day are needed to place a new box at the head. Block riffles are used at the head of the sluice followed by rails. Very little quicksilver is necessary, and it is found that no gold is recovered beyond 250 feet of boxes. Undercurrents are not used. Bowlders are removed from the pit by flat hand cars running on a steel track. If too large for this, which is seldom the case, they are plastered and shot with 75 per cent dynamite. The company is now in good shape to operate for a period of years, and estimates that the cost will not exceed 40 per cent of the product.

The company leases a portion of its ground where the situation is not advantageous for hydraulicking, and this is drifted. Although the drifting operations are carried on under great difficulties and require close timbering, the work is said to have been profitable.

In the hydraulic mining of the "White Channel" bench gravels of the Klondike, the small amount of water available, generally from 100 to 200 miner's inches, necessitates steep grades to the sluices. The gold is recovered with comparative ease because the bench gravels, after being thawed, are easily disintegrated and washed, and clay is absent from the pay streaks. The sluice shown on Pl. XXIV, *A*, is carried in a cut in the outer rim rock, on a grade of 12 inches to the box length of 12 feet. Twelve to fourteen boxes are ordinarily used. The details of the sluice are as follows: 24 inches wide by 20 inches deep, bottom of 2-inch and sides and lining of 1½-inch lumber; posts and sills 2 inches square; sills set in bed rock; riffles of spruce blocks 5 inches high and 9 inches square. These cost 25 cents apiece and last one season. Through this sluice 1,000 cubic yards are washed in twenty-four hours with 250 miner's inches of water. It is stated that 90 per cent of the gold is saved in the upper 6 boxes, only 1 per cent of the total gold being lost. The gold is mostly coarse and rough, but very fine gold is found.

In the case of another operation working the Bonanza Creek benche it was stated that gold was found the entire length of a 2,000-foot bed-rock sluice, followed by 500 feet of 2-foot wooden sluice, on grade of 11 inches to the box length. Block riffles were used in the wooden sluice. A small nugget was picked up 2,000 feet from the head of the sluice. Another operator, using 360 feet of 20-inch sluice boxes, on 12-inch grade, with iron-shod pole riffles, said that 2 per cent of his gold was escaping, and he was of the opinion that the installation of an undercurrent would pay.

In a hydraulic operation using 500 feet of boxes, 2 feet wide, 12-inch grade, with block riffles, and working a portion of the "White Channel" gravels where the gold is very fine, bright, and smooth, it was said that 80 per cent was caught in the top box, 85 per cent inside the first 15 boxes, and that none was found in the two end boxes.

In a bench hydraulic operation on Last Chance Creek, Klondike, a small undercurrent with cocoa-matting riffles had been put in after 400 feet of 2-foot sluices, with 12-inch grade, using block riffles. The undercurrent had given very satisfactory results, and a larger one was being constructed in addition.[a]

The above data indicate that where hydraulicking is possible in the Klondike, the gold-saving appliances are fairly adequate to the needs of the operations. As all hydraulic mining in that region is limited to ancient channels lying at an elevation of 200 feet or more above the present streams, water is exceedingly scarce, and sluicing methods must be adapted to this condition. These methods also depend on the necessity of impounding tailings on steep hillsides (see Pl. XXV, *B*), the high cost of labor ($7.50 a day), and of lumber ($80 per thousand feet), the necessity of making quick and frequent clean-ups, and the different economic ratio which obtains in recovering gold from small patches of rich gravel as distinguished from large bodies of low-grade gravel. When it is well established, therefore, that no great percentage of gold is escaping, it is likely that the extra cost of installing and maintaining complicated saving appliances would overbalance the increase in gold recovered. The use of drops in the sluices, and lessening of grade, might, it is likely, result in an increased saving, with no extra expense of maintenance, except in the wear of riffles at certain points in the sluices.

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[a] See Bowie, A. J., jr.: A Practical Treatise on Hydraulic Mining in California, p. 231, for full description of undercurrents and the method of leading gravel to them from the main sluice.

A. SLUICE BELOW ELEVATOR, OPHIR CREEK.

B. RETAINING DAMS, BONANZA CREEK, KLONDIKE.

In hydraulic elevator practice as pursued in various parts of Seward Peninsula, Alaska, short sluices are in use. Here, as elsewhere, a sluice is generally set in bed rock on a grade of 10 inches to 12 feet, or near this grade. On Ophir Creek, in the Council district, Alaska, the practice is to use from 10 to 12 boxes of 12 feet length, 24 inches wide and 16 inches deep, with wooden rail or pole riffles, shod with strap iron. This leads to the elevator sump, 10 feet deep. The position of the elevator with reference to the sluice below it is shown in Pl. XXV, A. The tail sluice following the elevator in the case specified is 28 feet higher than the head sluice. The tail sluice consists of five 24-foot boxes, 4 feet wide, and the practice is to give the first box either no grade, or a grade not exceeding 2 inches. The other boxes are graded from 2 to 5 inches successively, a 5 inch grade being retained for all below the fifth box, if more are added. On Ophir Creek, where there are six elevator plants in operation, iron riffles are generally used in the discharge sluice. A type of riffle which is used in the head box is made of railroad iron, cut in short pieces and set transversely in the box, forming a Hungarian riffle. Another kind in much favor throughout Seward Peninsula is composed of a set of T rails, in sets of 4 laid lengthwise (see fig. 32). These are made with a locking attachment, and are put in the sluice upside down. T angle-iron riffles, made in 1-foot square castings, laid in the sluice so that the slots set longitudinally, and with the flat side uppermost, are also very satisfactory and durable, and on account of their lightness facilitate the clean-up.

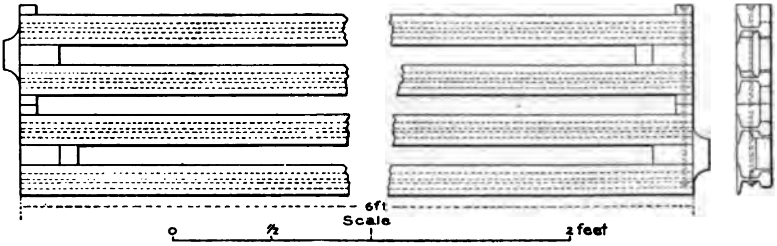


FIG. 32.—Rail riffles used in Seward Peninsula.

The proportion of gold caught in the head and tail sluices varies according to the practice. At Claim No. 2, below Glacier Creek, the gravel is run through 250 feet of iron boxes, only the two lower of which, next the elevator sump, contain riffles. These iron boxes are 10 feet long, 31 inches wide at the top, 27 inches at the bottom, and 8 inches deep, of a single bent sheet of iron, lapping 2 inches at the ends and fastened together with two bolts. The material slides easily through the cut in these, which are laid on 6-inch grade to the elevator. It is said that 50 per cent of the gold is caught in the last two of these, and 50 per cent is caught in the tail sluice after the elevator.

In the elevator operations on Ophir Creek and on Basin Creek, in the Nome district, it is said that 75 per cent of the gold is caught before the elevator, and in one case no gold was caught below the first four boxes of the tail flume. The experience in Seward Peninsula has shown that the gold is for the most part rough, and even crystalline, and even when finely divided is saved with ease. One marked exception was seen at Anvil Creek, where a 500-foot string of boxes on 4½-inch grade caught gold for the entire length.

## TAILINGS.

It has been said that the most important part of the equipment of a hydraulic mine is the dump. It has been found that the amount of material which can be moved is governed primarily by the grade of the sluice boxes. Reference to table 11 (p. 139), shows that there is a fairly constant ratio between the grade of the tail sluice and the amount of gravel moved through it per twenty-four hours per miner's inch of water used.

Since it is a prime necessity in hydraulic mining operations that as large an amount of gravel as possible shall be delivered through the tail sluice and disposed of on the dump, it is evidently of the highest importance that a steep grade, 12 inches in 12 feet if possible, and on no account less than 6 inches to 12 feet, should be available for the sluice. It is manifestly impossible to secure this grade if the natural slope of the ground or creek bottom over which the material is run is less than that required for the sluices. The grade of the surface must in fact be more than that of the sluice, in order that there may be ample vertical space below the end of the lowest box for the tailings.

In hydraulic operations on bench gravels, as, for example, in the Klondike region, the conditions for obtaining grade for the sluices are more favorable than in any other hydraulic operation of the North. The gravels of the Klondike White Channel benches are easily moved after they are thawed. This is fortunate, as on account of the small amount of water available hydraulic mining on the benches would otherwise be impossible. The expense of keeping up dams for impounding tailings, however, is considerable. Naturally the barren tailings, consisting of white quartz stones of the size of paving stones, can not be allowed to wash and slide down to the creek bottom. Were this permitted, the creek miners would soon find their cuts flooded and their ground covered with thousands of tons of useless débris. Therefore impounding dams for tailings must be constructed of the strongest available materials, in order to resist the weight of the stones. The cost of building tailings dams of brush and poles, 20 feet high, in the Klondike, is $5 per linear foot. Retaining dams along Bonanza Creek in the Klondike are shown on Pl. XXV, *B*.

In California hydraulic mines the dump, even after a series of drop-offs 25 feet in height, was sometimes in a canyon hundreds of feet vertically below the end of the sluice. The remarkable topographic conditions which allowed of such a phenomenal dump probably can not be found in another placer country, and certainly not in those portions of Alaska here considered. Even in California, however, it was often found necessary to distribute the dump by means of Y's near the end of the tail sluice. One such system is in practice on McKee Creek, Atlin, and has already been referred to. In hydraulic operations where 1,000 cubic yards or more a day are handled with low dump two carpenters have frequently to be employed at least half of the time in adding boxes to the tail sluice. If the dump be properly spread in delta form, the operation can be carried on in places where otherwise it might be impossible.

Where hydraulic operations are conducted in creek beds of low grade, below the so-called "sluice-box grade," or 6 inches in 12 feet, some expedient must be adopted to get rid of the accumulation of tailings below the end of the sluice. If the topography is such that the water runs off and only the solid residue of the tailings accumulates, the expedient is sometimes adopted of "piping" the dump away with an extra giant set in a convenient position for the work. This giant generally requires the services of one man, and, although used only a portion of the time, may be figured under Alaska conditions to add from 2 to 5 cents to the cost of handling the gravel. The efficiency of the plant is detracted from by so much power as is required to push the tailings out of the way. It can not be too strongly insisted on that when this or any other device is used to move or elevate the tailings, the strict principle of hydraulic mining is departed from. The use of the steam or horse scraper in removing the tailings of small plants is discussed on pages 70–72.

If the grade of the creek bed is lower than that necessary for the sluice, the tailings must be elevated and the water pumped from the pit. Mechanical elevators with the addition of a bed-rock drain are sometimes used, though not with success, so far as known. In one case on Mastodon Creek of the Birch Creek district the installation of a steam scraper for lifting the tailings from the hydraulic pit was contemplated, the water being handled by a bed-rock drain, which had been installed at great expense.

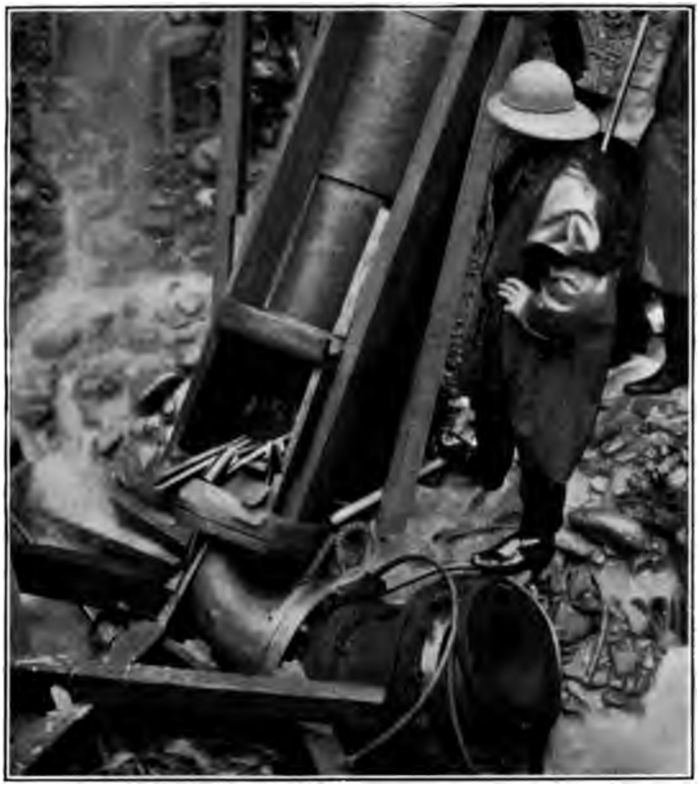
## THE HYDRAULIC ELEVATOR.

The hydraulic elevator is too well known to need elaborate description. The principle is the same as that of the injector. Water under pressure discharged through a nozzle set within a steel jacket creates a vacuum and causes water—about one-half the quantity used by the elevator—and accompanying solid material to rise to a height corresponding to approximately 10 per cent of the head under which the water acts. Many elevators have been used, and they all act on the same principle. One of the necessities is the constant admission of air with the tailings, which, by being compressed, aids in the elevating process. Those elevators are considered best which admit air most freely.

An exceedingly simple form of hydraulic elevator—the so-called "open" elevator—is in use in southern Oregon. The gravel is piped directly into an inclined box 8 by 30 feet in dimensions, which has a false bottom of punched iron plates. All the fines run back on the plank floor of the box to the undercurrent, the oversize falling to the tailings pile at the raised end of the box. The contrivance is a makeshift, but is said to be successfully used in the flat stream beds of southern Oregon.

A hydraulic elevator will sink a pit in a flat piece of ground containing no large stones. In starting new work on placer ground it is, in fact, the practice to sink the pit with the elevator. The portion of the hydraulic elevator shown in Pl. XXVI, *A*, is in position in the bottom of a pit 40 feet deep, on Glacier Creek, Seward Peninsula. This pit, which is 90 by 60 feet in dimensions, was sunk in ten days by a 7-inch elevator, smaller than the one shown in the picture, using 325 miner's inches of water at 360 feet head. At the claim next higher up the creek a similar elevator was in use, and a description of its operation will be of interest. Seven hundred and sixty miner's inches, under 330 feet pressure, were used in the operations. The water is afforded by the Miocene ditch, described on pages 123–126. The grade of Glacier Creek does not exceed 50 feet to the mile in its lower portion, where this property is worked. The gravel deposit, which is wide and 20 feet in depth to bed rock (see Pl. XXVI, *B*), can not be worked by the hydraulic method on account of the impossibility of disposing of the tailings. Therefore a long cut is ground-sluiced off to the depth of 6 or 7 feet below the surface. The elevator sump, which must be about 5 by 5 feet and 10 feet into the bed rock, is sunk in the bottom of the large pit. The elevator, which, with the head above given, lifts 36 feet, is set in position. For this ground a No. 1 elevator, having a 10-inch throat and using 4½-inch nozzle, is employed.

The hydraulic elevator, like the one shown in the photograph, is by no means simple. It consists of 15 large parts, some of them very heavy, and a number of rods, bolts, bands, and wooden staves. The cost of the one shown is approximately $750 in San Francisco, and the weight is 2,000 pounds. A pipe is led from the supply pipe to feed the nozzle of the elevator. It is connected to this by means of the ball joint. This joint, shown in the photograph, is exceedingly convenient, as it allows the pipe to enter the pit from one of several directions. Where the upcast pipe joins the top casting of the elevator it



*A*. HYDRAULIC ELEVATOR IN PIT, GLACIER CREEK.

*B*. GRAVEL IN PIT, GLACIER CREEK.

should be made to fit over and outside of this casting, so that there will be no leak. The casting should be rimmed outside to a depth of 2 inches. The jacketing of the nozzle is considered very good practice, as it prevents smothering with gravel. In the type of elevator here shown the advantage is claimed that the various parts, lining, backing, first section above throat, top section, etc., are separate and merely fit together, being held by a ring of wooden staves, as shown in Pl. XXV, *A*. This obviates the use of heavy castings. The upcast pipe is made of lap-welded steel, one-eighth inch in thickness, and either lead or flange joints can be used. In the plant described the pipe was in 18½-foot lengths and the joint was made by bolting together wooden blocks, 10 by 12 by 24 inches, with a half-round cut made to admit the pipe. The upcast pipe should be set at an angle of 60°.

By far the most important parts of the elevator as regards wear are the throat and the hood which receives the impact of the gravel in the head box of the tail sluice above. These two parts are manganese steel castings, and if the gravel is very sharp and siliceous may wear out in one season. For the 10-inch elevator the throat weighs 400 pounds and the hood 600 pounds (price 16 cents per pound in San Francisco). The manner of setting the hood in the head box of the tail sluice is shown in fig. 33. The upcast pipe is laid against an inclined framework of four lengths of 2-inch plank 2 feet wide, and extends directly into the bottom of the tail sluice. (See Pl. XXVII, *A*.) Leading to the throat of the elevator directly above the nozzle is a sluice box, which conducts the gravel either from the boxes preceding it or from a bed-rock sluice, which is carried back on grade in the pit bottom as fast as the gravel is piped down. On Glacier Creek the gravel is led to the elevator in 24-inch iron boxes on 6-inch grade, with or without rail riffles. After leaving the discharge of the upcast pipe the gravel passes through eight or nine 36-inch boxes, with T-rail riffles. The first

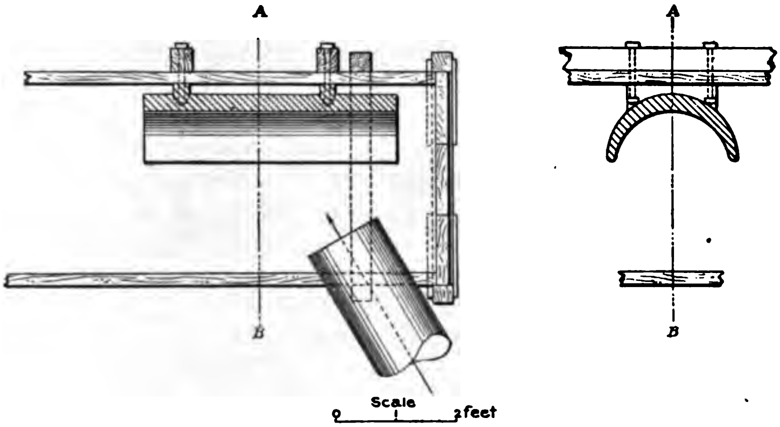


FIG. 33.—Hood in head box of tail sluice.

two boxes have 4-foot sides, reinforced inside and out with sheet iron, and are set high on a trestle to give the desired elevation for maintaining the dump. The grade is either 0 or 1 inch in 12 feet for the head box, 2 inches for the second, 3 inches for the third, 4 inches for the fourth, and 5 inches for the succeeding boxes. It is proposed to supplant wooden sluice boxes on this claim with iron boxes 8 to 10 feet long and from 2 to 3 feet wide, of one-fourth-inch sheet iron, weighing from 500 to 600 pounds each and costing from $15 to $20 in San Francisco. Five hundred and twenty miner's inches of water are used for the elevator and 240 inches are used through the 3-inch nozzle of the giant. The water is distributed from the supply pipe by 11-inch pipe to the two nozzles. By this arrangement 500 cubic yards of gravel can be handled in ten hours. It is said that the elevator is not handling over one-half the gravel that could be put through it if the larger amount could be moved to the throat. Experiments with scrapers operated by water power are in progress.

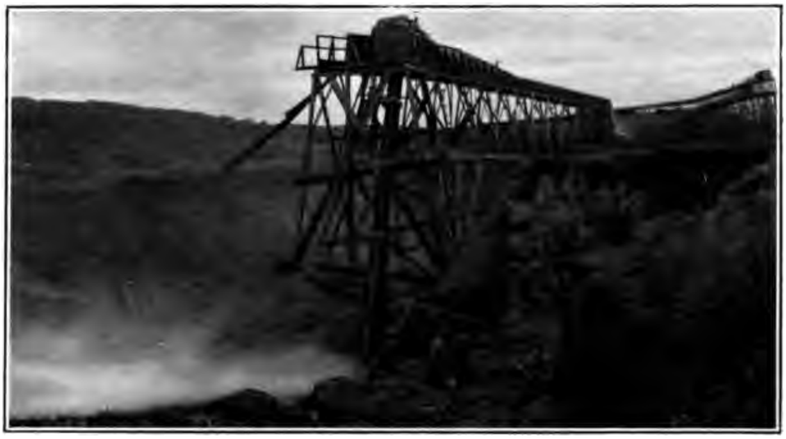
The practice here is first to run back with a narrow cut 250 feet up the creek from the elevator throat, then to come forward toward the elevator, breaking down the sides.

If the cut is 150 feet wide, 250 feet long, and 25 feet deep, a new set-up must be made every thirty-four days, assuming that the plant is run all the time at its full capacity. To make the new set-up of boxes and to build the trestle requires $400 worth of lumber and the labor of 5 men for five days. In addition, it takes the whole crew of the placer one day to take the elevator from its old position and place it again. The total extra expense is, then, nearly $1,000 in money and at least one full day out of the short working season each time that the elevator is moved.

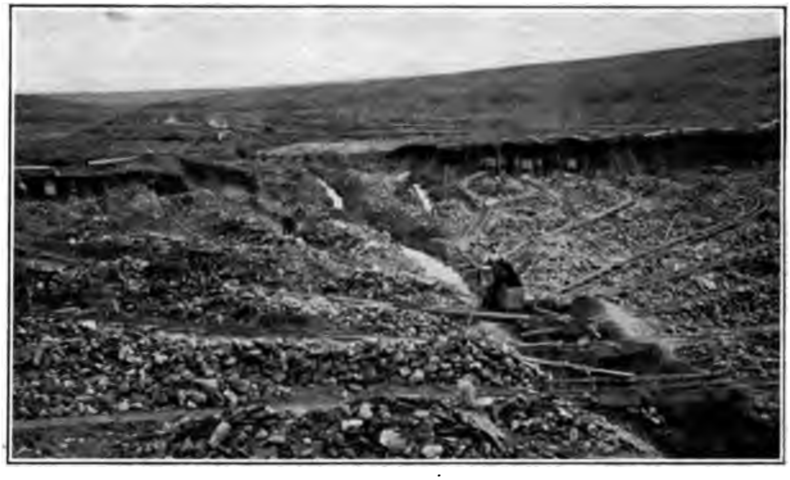
In Seward Peninsula the handling of bowlders gives little trouble with the elevator. A 10-inch throat will take a 9¾-inch bowlder. It is not uncommon to see the men near at hand throwing small stones down at the throat of the elevator. This is because stones are jammed in the throat. The elevator for the moment is not clearing the pit, and the small stones artificially fed will act as small cannon balls to break the obstruction.

It should be noted that a very large amount of water is used for the elevator as compared with that used for the giant. It has been found most economical to distribute the water one-third to the giant and two-thirds to the elevator. If the elevator is too large for the water used, the employment of a choked nozzle or smaller nozzle will obviate the difficulty, while the larger throat does no harm. One of the most common difficulties with the hydraulic lift is the fact that it will choke if required to raise too much seepage or pipe water.

Elevator practice in hydraulic mining has been so limited in its application, and has met with so little success, that definite rules for



*A.* HYDRAULIC ELEVATOR.



*B.* OPEN CUT WORKED BY TRAMMING TO BED-ROCK SLUICES.

the guidance of the miner have not been formulated. The best that can be done is to cite examples of the results of actual practice and to enumerate the difficulties experienced. It is a common saying among users of hydraulic elevators that the machines will handle all the gravel that can be got to the throat.

The table for use in hydraulic-elevator operations is as given by the manufacturers:

TABLE 12.—*Water required in hydraulic elevators.*

| Number. | Water required. | Size of nozzle. | Size of throat. |
|---|---|---|---|
| | *Miner's inches.* | *Inches.* | *Inches.* |
| 1 | 300 to 500 | 3½ to 4½ | 10 |
| 2 | 500 to 750 | 4½ to 5 | 12 |
| 3 | 750 to 1,000 | 5 to 5¾ | 15 |

The above heads are those which are generally available in Alaska. The amount of water that can be handled successfully, in addition to that coming through the nozzle, does not in practice exceed one-third. If there is a large quantity of seepage water the excess must be taken care of by means of an auxiliary water lift. One of these is shown in position in Pl. XXV, *A*. It takes an 8-inch supply pipe, choked to 2-inch nozzle in 4-inch throat, with 70 inches of water at 100-foot head, to lift 80 inches of water 26 feet. This lift weighs 350 pounds and costs $140 in San Francisco. Whether such a lift is needed for continuous operation or not, it is convenient to have it at hand for assistance in temporary pumping of water from the pit during the time of clean-up and other periods when the main elevator is not acting. Double elevators are sometimes used in preference—a much more expensive installation.

All the hydraulic elevators seen in Alaska were of small capacity, carrying 10-inch throat or less. It should be remembered that the tables and estimates of percentage of lift to head are generally based on the assumption that the throat piece of the uptake pipe is double the diameter of the nozzle employed. In northern practice it is generally the experience that a smaller nozzle must be used than the theoretical size, and in consequence the lift is decreased.

The type of plant illustrated in Pl. XXVII, *B*, is not advisable except as a necessity, where surplus water from a head ditch is available. The gravel is shoveled from the cut by hand into cars which are trammed and dumped into the bed-rock sluice running to the elevator. The operation has no relation to hydraulic mining proper. The plant figured has a capacity of 700 cubic yards in twenty-four hours, the working expenses being no less than on a neighboring claim, with almost precisely similar geological conditions, where gravel was being handled by shoveling into 1¼-yard derrick skips trammed on trucks to the derrick, lifted to a sluice, and washed with gravity water. Now, grant that the cost of the derrick plant was $10,000 while that of the elevator plant was only $5,000. The water for the elevator, however, 500 miner's inches, represented one-fifth of that obtained by an elaborate ditch system, the proportionate charge of installation of which was $60,000. It is very unlikely that the life of either of these properties will exceed six years, and it can be easily seen that the amortization charge for cost of plant must be, less interest allowance, $17,000 in the elevator plant, as against $10,000 in the derrick plant.

It would be unfair to entirely decry the use of the hydraulic elevator in Alaska. On the other hand, it is a deplorable spectacle to see cases where expenditures of from $25,000 to $200,000 have been made to secure the use of water at a head, where the elevator is installed, and where it finally becomes apparent that loss will result in the operations because gravel can not be moved to the elevator in sufficient quantity.

To take a very typical Alaskan case, assume a body of material which, handled at the rate of 700 cubic yards a day, can be worked out in six seasons; that the section consists of 2 yards of muck and of 1½ yards of gravel which has a tenor of $3 per cubic yard. This material can be all handled by the hydraulic method at an expense of 50 cents per cubic yard, actual working cost, including superintendence. In order to make this hydraulic method available a ditch carrying 1,000 miner's inches of water at lowest stage must be built at a cost of $100,000. Let this cost include the hydraulic equipment. Simple interest charge on the investment and maintenance of ditch and plant will amount to $90,000 additional. Allowing nothing for purchase of property, the five annual payments to amortization of plant fund will amount each to $38,000. Allow one hundred days each season actual working time. Then the amount per cubic yard which must be added to cover payment to the sinking fund will be 0.542 cents per cubic yard, or a total of $1.042.

This cost will not be regarded as extreme by those who have had experience in similar undertakings. The operator has a choice of mechanical plants, by using one of which he could work the ground with great economy. In other words, the use of the hydraulic elevator in Alaska frequently defeats the object for the attainment of which the hydraulic method of mining is employed.

## DREDGING.

### INTRODUCTION.

The opportunities for gold dredging in the far Northwest are not by any means numerous. Perhaps no part of the gold-mining industry is dependent on so many conditions for its financial success as the dredging of auriferous gravel deposits for gold. Some years ago, in connection with Mr. J. B. Landfield,[a] I published a review, setting forth the geological conditions which are necessary for the success of gold dredging, the consideration of which is frequently neglected by those contemplating the installation of dredges. The portions of that paper which deal with the present case are here quoted. The prime consideration is that conditions of a peculiar character are a necessary accompaniment of the values in any locality where it is proposed to win gold by dredging. Recent experience has proved that the greatest attention must be given to topographic and geologic conditions in the country in which the enterprise is to be inaugurated, otherwise success in the operations is not to be looked for. The following explanation of the conditions necessary for dredging is therefore pertinent:

> Taking the country about Oroville, Cal., as the best example in the United States, let the geological conditions be considered. To the north and east the erosion of a vast extent of mining country, whose rocks are penetrated by gold-bearing veins, has contributed little by little through geologic ages to the mass of detritus now occupying the bed of Feather River. The wearing down of mountains, originally very much higher than at present, through a vast amount of time has caused the formation of a valley of extraordinary width but of no great depth. The massing of stream detritus is also responsible for a decrease in the gradient and a slowing down almost tö a topographical equilibrium of the formerly swift current of the river. The stream, unable under these conditions to cut its way by virtue of the material which it carried in suspension, has for a long time been depositing its load, filling its wide valleys with sand and gravel, together with the less destructible of the metallic particles, and notably gold. Geologically speaking, the rate of deposition of the river's material may be said to be on the increase and the current of the water, still of considerable velocity, to be greatly lessening in swiftness. Later in its geologic cycle Feather River will doubtless assume, on a smaller scale, the present character of the Mississippi, forming oxbow curves, cut-offs, and, as it were, losing its way among constantly shifting sand bars.
>
> The accompanying residual gold, as it travels to a greater and greater distance from its original source in the veins of the mountains, becomes more and more finely divided, even to microscopic dimensions, and increases in purity, and the degree of evenness with which the particles are distributed in the gravel becomes a phenomenon of constantly increasing definiteness and importance.
>
> Such a set of conditions as that obtaining on Feather River is the exception rather than the rule in the western part of the United States, and even in California. The above explanation has been entered into in order to present in some measure the reasons why, on the California river referred to, the conditions are not only favorable but eminently suited to gold dredging. Shallow gravels—that is, those less than

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[a] Purington, C. W., and Landfield, J. B., jr., The gold-dredging fields of eastern Russia: Eng. Mag., vol. 22, 1901, pp. 398-407.

> 40 feet in depth—are an essential in dredging enterprises.[a] Thus broad rather than narrow, gorge-like valleys must be sought. Bed rock of a soft, decomposed character, which can be easily cut into by the dipper or bucket of the dredge, is a necessity, since placer gold, by virtue of its gravity, often sinks from 1 to 2 feet into the crevices of the rock underlying. It is hardly necessary to state that the amount of gold increases in geometrical ratio the lower it lies in a given bed of gravel. Thus old rather than young valleys are favorable for dredging.
>
> Additional reasons of great weight why geologically old valleys should be looked for are that the size of bowlders is greatly decreased, gravel becomes by long abrasion uniform in size, the angularity of the fragments disappears, and a bed of pebbles, round and easily handled, is the result.
>
> The even distribution of the gold which, as mentioned above, is an invariable accompaniment of old and wide valleys, is a point in favor of this sort of mining, looked at from the standpoint of a business enterprise. At the same time it is evident that the finely divided state in which such gold is found necessitates the highest skill in recovering it. For example, at Oroville, although some of the operators are saving gold the subdivision of whose particles is almost microscopic, it is thought with good reason that a considerable percentage of values is lost in the water used for the washing.
>
> The conditions on Yuba River in California are, from the dredging standpoint, as favorable as those at Oroville.

The conditions above mentioned as important to the success of gold dredging should be taken account of as much by the men who contemplate installing dredges in Alaska, as in any other part of the world. While it is true that there are in Alaska broad valleys, such as those of Yukon River and its larger tributaries, and that they have been found to be to a certain extent gold bearing, yet peculiar and formidable conditions serve as a barrier to the exploitation of these streams by dredging. A remarkable illustration of this fact came to my notice during the last summer. In a small island facing the settlement of Eagle, in the interior Yukon country, a shaft sunk through rolled stream gravel to a depth of about 40 feet in winter penetrated perpetually frozen ground until near the limit of its depth. At this point an underflow of water was encountered so strong as to prevent further prospecting. It was said that good values in gold, even up to $1 to the yard, were found in sinking this shaft. If this be true it would imply that considerable areas of the underlying immensely wide gravel flats of Yukon River are gold bearing. Under other conditions portions of the ground might be found workable by gold dredges. Experience has shown that in Alaska the permanent frost which exists in much of the river country renders the exploitation of the gravel by ordinary mechanical means absolutely impossible. It is the opinion of those who have attempted the dredging of frozen gravel in the Klondike that, while the devising of a dredge strong enough to dig frozen gravel is not impossible, the undertaking has not yet reached even the experimental stage. In the interior the cost of dredging unfrozen ground is 40 cents per cubic yard, while the dredging of frozen ground by thawing ahead of the dredge with steam points requires an expenditure of 40 cents additional. It will be recognized by those familiar with dredging that such a cost is prohibitive of the exploitation of the average dredging gravels.

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[a] Since the above was written the enlarging and strengthening of dredges has resulted in successful digging (on Yuba River, California) to the depth of 60 feet. It should be noted, however, that the upper 25 feet of the Yuba gravel section consists of loose tailings.

River-bed gravels seldom average over 20 cents to the cubic yard in gold, locally enriched area from side tributaries necessarily being excluded. It has been frequently stated by engineers that the average tenor of the Oroville gravels is not above 15 cents to the cubic yard. I see no reason to believe that the gravels of the Alaska rivers, the Yukon, Tanana, Koyukuk, Kuskokwim, and their larger tributaries should be any higher. It is likely that only in very restricted areas will the tenor in gold of these streams be as high as 15 cents. It should be distinctly understood that Solomon River, Snake River, and streams of similar size in Seward Peninsula are not properly called dredging streams. They do not by any means fulfill the conditions implied in the foregoing geological description. While it is undeniable that in certain isolated cases streams having the character of Solomon River may be profitably dredged, the enterprise must be considered as lying outside the legitimate province of the gold-dredging industry as developed in California and New Zealand.

A recent study of the dredging fields of California has led to the conclusion that while dredging is possible to a greater depth than formerly (60 feet in Yuba River), there has been little reduction in the general costs of work. Eight cents per cubic yard is the lowest cost which may be safely counted on at Oroville, the principal field.

No special devices have been put into practice for breaking up cemented gravel, and some of the dredgers at Oroville are compelled to use powder to shake the bank in front of the dredge. Such practice would have no beneficial effect in handling frozen gravel. The use of hydraulic nozzles attached to the ladder frame at the lower tumbler seat has several times been suggested by engineers for assistance in thawing frozen gravel as the dredge operates.[a] Experience in hydraulicking frozen bench gravels in Alaska has proved that such a device would be of no avail, since it is mainly the action of the sun and not that of the water which thaws the gravel face. The possible method of steam thawing is to drive long pipes from the surface into the ground ahead of the dredge (the practice in Bonanza Creek in 14-foot ground). The cost of this would probably be prohibitive when the depth of the ground exceeds 15 feet.

The only manner in which heat can be applied directly to the part of the gravel face where the buckets are digging is by means of elec trically heated water thrown against the bank. The fact that hydraulicking with water heated to 50° and applied directly to the face through a nozzle in Klondike drifting operations is remarkably effective in thawing frozen gravel suggests possibilities in regard to dredging. This possibility is extremely remote, but might under certain circumstances be applicable where the dredge is operated by electric power generated by water under head.

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[a] Levat, E. D., L'Or en Sibérie orientale.

The fact the dredge is the only appliance used in placer mining which transports its sluices and continually provides new room for its tailings renders it attractive in the consideration of the shallow northern placers.

In Seward Peninsula there are some river deposits which contain gold sporadically distributed and in which frozen and unfrozen areas alternate. Extensive prospecting has resulted in the discovery of a certain amount of ground which appears to warrant the installation of dredges of special design. More of these areas may be found, and it is probable that some of them may be in the interior. It is not believed that dredges can be operated successfully in any of the area included in the South Coast provinces, as there the geological conditions are almost the exact opposite of those outlined at the beginning of this section.

### DESCRIPTION OF OPERATIONS.

Notwithstanding the difficulties encountered in the interior of Alaska, the possibility of dredging in this field under certain favorable conditions is not to be denied. A small dredge for prospecting purposes was being operated cheaply in Stewart River in the Yukon Territory, not far from the international boundary, during the season of 1904. It subsequently operated in Klondike River. This dredge has 39 open connected 2¼-foot buckets, and handles 750 cubic yards in twenty-four hours. It digs to 30 feet, and in Stewart River very little frozen gravel was encountered. A crew of 6 men is employed and the daily working cost alone is said to be 7 cents a cubic yard. The hull was built at White Horse, of 4-inch planks of imported Vancouver fir, and cost $5,000.

A complete description of this dredge has been published by the designer, Mr. A. W. Robinson.[a]

A dredging installation was also visited on Gold Run, in the Atlin district of British Columbia. The power plant was situated on Pine Creek, 5 miles below the dredge. A flume and ditch line 1½ miles in length supplied 1,500 inches of water, at 187 feet head, to 2 Victor turbine wheels of 500 horsepower each. It was said that 80 per cent of this efficiency was available at the dredge. The current was delivered at 1,100 volts to the transformer and this was stepped up to

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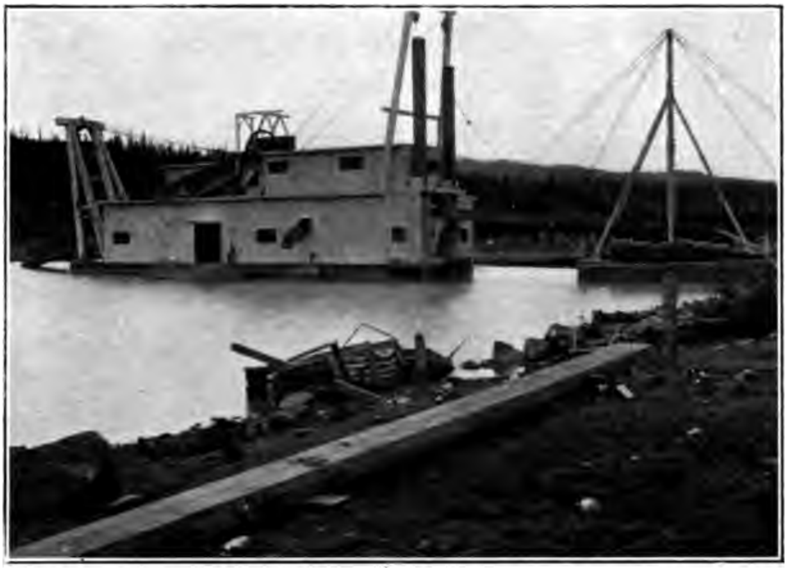
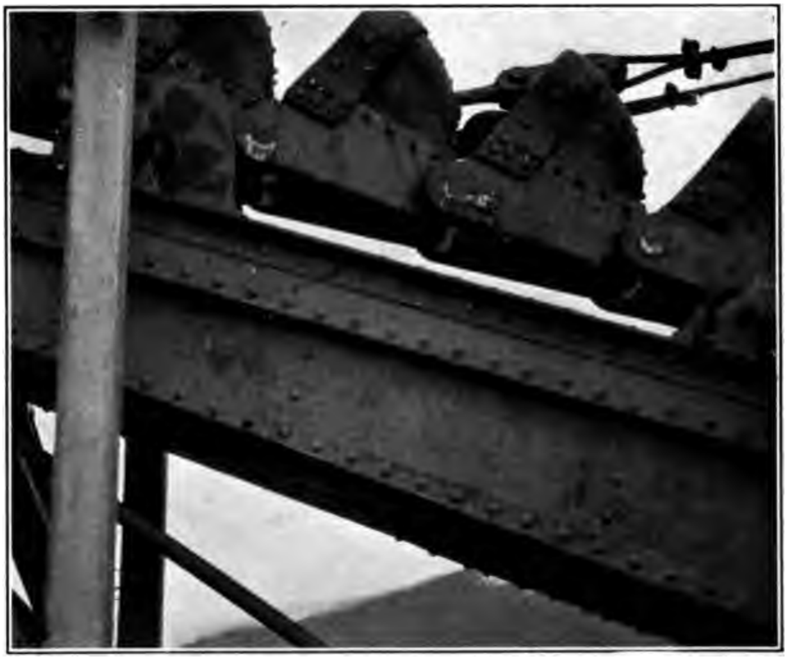
[a] The Stewart River gold dredge: Canadian Mining Institute, March 4, 1903.



*A.* SINGLE-LIFT DREDGE, 40-FOOT GROUND AT ATLIN, COARSE GOLD.



*B.* BUCKETS, ATLIN DREDGE.

22,000 volts before its conduit over No. 6 copper wire 3-phase system to the dredge transformer, where a step-down to 500 volts was made. The turbines have 24-inch intake and 30-inch discharge, and the gates weigh 2½ tons each. The water is available from April 1 to November 15. The power plant cost $50,000, not including an outlay of $10,000 for a 5-mile pole line, 3 copper wires, and 20 telephone wires.

The pole line was built by contract requiring that the poles should be 6 inches in diameter at top, 24 feet above and 5 feet below the ground, and that timber should be cut for a width of 40 feet, all at a cost of $2,250 for 7 miles.

A general view of the dredge is shown in Pl. XXVIII, *A*. The cost, installed on the ground, was $200,000. It is of the close-connected, 3-foot bucket, single-lift type, arranged with trommel and pivoted sluice box, and tables carried on separate float. This type of dredge was considered necessary on account of the presence of coarse gold in the deposit. That by all dredging engineers the occurrence of coarse gold is not thought prohibitive of the use of the stacker dredge is apparent from Mr. Hutchins's remarks on this subject (see pp. 173–190). The arranging of the entire tail sluice to turn in an arc, operated by stern lines, permits the distribution of the dump. The system of digging is against a spud and not a headline. Operations had only begun when the dredge was visited; its estimated capacity is 2,500 cubic yards in twenty-four hours, digging to an average depth of 44 feet. The dredge is equipped with 5 motors, and a total of 200 horsepower is used. The total weight is 400 tons.

The hull was built in a pit on the ground, of Vancouver fir, and afterwards floated in the pond shown. It has a draft, loaded, of 3½ feet, and is 90 by 36 feet in dimensions.

The winches are controlled by levers directly above in the pilot house on the starboard side of the upper deck. The 7 drums operate the 4 side lines, main ladder line, and 2 spud lines.

The digging ladder is 90 feet long, 3 feet wide and 2 feet deep in the center, and is braced by steel trusses. The tumblers are of pentagonal form and of steel, with manganese steel plates riveted on, with 8-inch shaft. There are 96 buckets that hold 3 cubic feet each, have the shape shown in Pl. XXVIII, *B*, and are equipped with manganese steel lips, pins, and bushings. As shown by Pl. XXVIII, *A*, a bowlder weighing 1,500 pounds rode one of the buckets and was taken off on the upper deck. The buckets will dig 40 feet in front of the bow, and a bank 8 feet above water can be handled. The rate of wear in bucket lips had not been determined. A nozzle is used to clean the buckets as they pass the upper tumbler. No save-all is used. The sprocket wheels and chain are of steel, the diameter of the upper wheel being 10 feet.

The motors are of the following rated capacity:

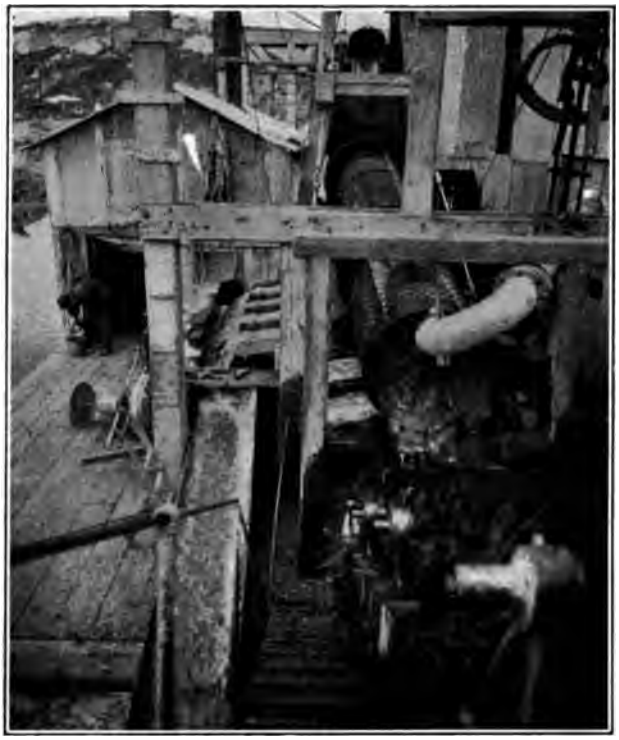
| | Horsepower. |
|---|---|
| For digging | 100 |
| For 10-inch centrifugal pump | 50 |
| For 6-inch centrifugal pump | 20 |
| For screen | 20 |
| For raising ladder and operating lines | 30 |

The gravel is raised 27 feet above the water. The trommel is 5 by 12 feet on 8 per cent grade, punched with holes that are 1½ inches in diameter in the upper plate, 4½ by 7 inches in the middle plate (enlarged from smaller size), and 4-inch holes in the lower plate. Thirty-five per cent of the material is oversize and is dumped directly from the end of the trommel into 2 side chutes overboard. The trommel has 6-inch spray pipe, supplied by the 6-inch pump with water under 40 pounds pressure. It is estimated that the trommel plates will last one season.

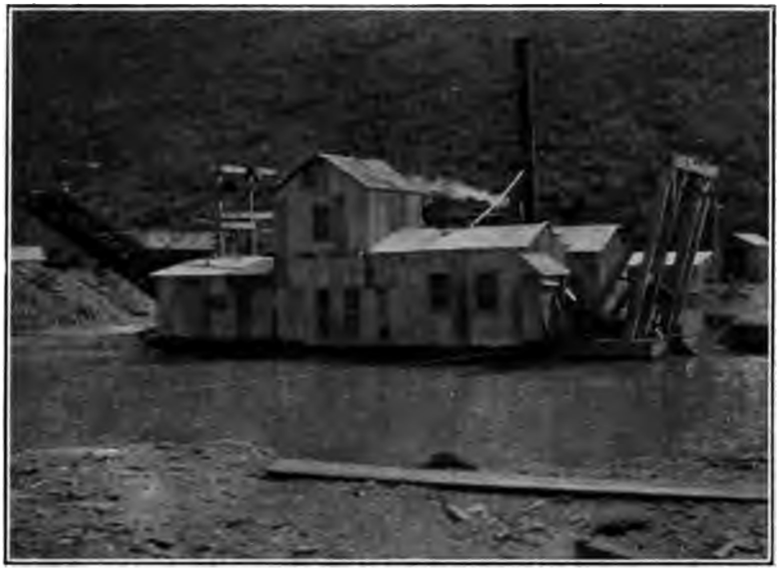
From the trommel the undersize falls directly to a sluice connected by means of a pivoted box to the main sluice in the auxiliary float. This sluice is 4 feet wide, 120 feet long, and 2 feet deep, is of steel, has a grade of 8 per cent, and is provided with Oroville Hungarian riffles and grizzlies to allow the material to pass to undercurrent tables, which have an area of 1,000 square feet (see p. 195). The dump of the main sluice is 8 feet above the water. Two hundred pounds of quicksilver are kept in the sluice and tables.

The cost of dredging at this plant is estimated at 8 cents per cubic yard for the capacity above given, only actual labor, supplies, and superintendence on the ground considered.

A dredge of the New Zealand type was operating on Bonanza Creek in the Klondike district. It is shown in Pl. XXIX, *B*. The boat had been in service for several years, having been first installed on Lewes River. It had then been towed and hauled by cables down Yukon River to the mouth of the Klondike and then taken apart and reerected on Bonanza Creek, where its operations have continued during the open months since July 1, 1903. A crew of 24 men operate the dredge and thaw the gravel in front of it. The thawing is done by means of 11-foot pieces of gas pipe, 12 in number, driven vertically into the ground. The bed rock here is a sericite-schist and the dredge is said to clean it well. The capacity of the dredge working under present conditions is 500 cubic yards in twenty-four hours. The bucket line is equipped with thirty 3½-foot buckets, open connected. The ladder is 50 feet in length and will dig 28 feet below the surface of the water. At present the dredge is digging to a depth of 14 feet and 8 feet in front of the bow. The stacking ladder carrying a steel-bucket conveyor is 38 feet long, and carries all material over 1½ inches in diameter, the size of the largest holes in the trommel. The lips of the digging buckets are of manganese steel and last four



*A.* TROMMEL AND GOLD-SAVING TABLES, NEW ZEALAND TYPE, BONANZA CREEK, KLONDIKE.



*B.* NEW ZEALAND DREDGE, BONANZA CREEK, KLONDIKE.

months in use. New buckets cost $22 landed. The dredge digs on a 1-inch headline with five-eighths inch side lines, no spuds being used, and appeared to dig the gravel very satisfactorily after it was thawed. There is not over 2 per cent of the gravel which is over 18 inches in diameter. The hull is 71 by 30 feet in dimension and draws 4 feet of water; 55,000 feet of Oregon fir was used in its construction. The bow and ladder gauntrees are 15 feet high and are built of timbers 12 inches square. The dredge is operated by steam power. One vertical engine of 50 horsepower operates the dredge exclusive of the winches, which employ a separate 10-horsepower engine. The boiler carries 110 pounds of steam and in twenty-four hours burns 3 cords of native spruce wood, which costs $10 a cord. The winches are on the starboard bow side, the winchman's position being on the lower deck.

Since the trommel was built its length has been increased and is now about 15 feet, with $\frac{9}{16}$-inch holes in the upper plate and 1½-inch holes in the lower. The trommel was extended so that none of the nuggets, which are worth from $12 to $200, would be lost. The trommel distributes the fines to 8 tables, 42 by 30 inches in dimensions (see Pl. XXIX, *A*), the distribution being regulated by iron gates. The tables, fitted with cocoa matting and expanded metal, have a grade of 1½ inches to the foot, and are followed by 2 sluice boxes 20 feet long by 2 feet wide, with angle-iron riffles, which discharge at the stern, as shown in the illustration. An inevitable difficulty with the dredge is the banking up of this material at the stern. Three thousand gallons of water per minute are pumped by a 10-inch centrifugal pump. The tables are cleaned up every day; sluices, once a month. The season for operating is from May 15 to October 1. It is understood that the dredge was successfully operated during the season of 1904 on ground which had been previously drifted.

The cost of the dredge laid down originally in Lewes River was $125,000.

A shift sheet used in this dredge, devised by Mr. J. M. Elmer, is here appended and may be of interest to those engaged in or contemplating dredging operations.

*Shift sheet of Lewes River Mining and Dredging Company.*

Date, ———.
Number of shift, ———.
Winchman, ———.

| Lost time. | Materials used. | Remarks. | Hours. | Minutes. |
|---|---|---|---|---|
| Ladder and bucket line | | | | |
| Stacker | | | | |
| Winches | | | | |
| Screen | | | | |
| Pumps | | | | |
| Lines | | | | |
| Clean ups | | | | |
| Oiling | | | | |
| Boiler | | | | |
| Condenser | | | | |
| Line shaft | | | | |
| Engine | | | | |

Character of ground, ———. Total lost time, ———. Depth of ground, feet, ———. Total running time, ———.

Foreman, ——— ———.

No dredging operations were in progress in the interior of Alaska during the summer of 1904.

In Seward Peninsula two dredges were operated in Solomon River and two land dredges were in operation in the vicinity of Nome, while another dredge was being constructed in Snake River near Nome, and a dredge was operating in a small way in Niukluk River, 1 mile above Council.

About one-fourth mile east of the town of Nome a small land dredge was operating on beach sand in the maritime strip. The section on which the dredge operated was 12 feet in thickness, and consisted of 6 feet of beach sand, 4 feet of clay, 2 feet of gravel, and 2 feet of underlying broken schist.

The dredge, which cost $5,500 in Seattle, was operated with a 6-horsepower gasoline engine. In addition, power was furnished to a centrifugal pump, which had the suction pipe led 60 feet out into the sea water. Eighteen gallons of gasoline, costing 40 cents a gallon, were burned in ten hours. The dredge was mounted on trucks surmounted by a turntable, allowing it to turn on a circular track. The average capacity was said to be 200 cubic yards a day. This represents one of the many attempts which have been made to dredge the Nome beach sands, none of which, so far as known, have met with any success. This material is unfrozen, but many difficulties are encountered. Storms which render the beach uninhabitable are causes of long delay, and the attempt to pump the sea water is a cause of constant trouble on account of the seaweed and wreckage which clogs the screen.

A small land dredge running on a 40-foot track, and digging a canal beneath for the purpose of laying a long bed-rock drain, was operating on Bourbon Creek, one-fourth mile north of Nome. The work was in an experimental stage, and a large dredge of the same type, of 500 cubic yards daily capacity, was under construction.

A floating dredge of the dipper type, with sluices arranged on a separate hull, was under construction in Snake River, one-half mile from Nome, but the data relating to it are not available for publication.

A small bucket dredger so arranged that the digging ladder can turn through an angle of 20° has been operating in Solomon River, 3 miles north from the coast for the last two seasons.

In the operations of 1903 a 60-foot width of the channel of Solomon River was worked and was found to be unfrozen; the gravel averaged 5 feet in thickness and rested on schist bed rock. The tenor is said to have been $1.13 per cubic yard. The greatest thickness encountered from the surface of water to bed rock was 12 feet. One and one-half feet of the schist bed rock was taken up by the dredge. The bed rock is generally soft, but has hard reefs. The largest stones found were 1 foot in diameter. The dredge was operated by 5 men on 2 shifts, the amount of running time being from twelve to twenty hours out of the twenty-four. The men are said to have made a profit, paying a royalty of 25 per cent on the output. The dredge is of 16-horsepower capacity and handles an average of 150 cubic yards in twenty-four hours. The operations consume two to three cases of gasoline a day, the average for fuel being $12.50. The total expense is figured at $30 a day. The experience with the pay is that the ground is very spotted, being in places rich and in places barren. There are 32 buckets of 1 cubic foot capacity on the ladder. A cut of 30 feet across is made in one setting. The ladder is made of 4-inch cast-iron pipe and can be lengthened by adding pipe and buckets. In 1904 the dredge was being operated by 2 men on a shift, one man running the winches, and one man attending to the engine and picking out the large bowlders and throwing them overboard. The bucket lips are made of tool steel.

The sluices into which the buckets dump directly are fitted with punched-iron screen of the "Caribou" pattern, described elsewhere, followed by Hungarian riffles. The sluice is 64 feet in length, 23 inches wide, and 12 inches deep, with a grade of 15 inches in 12 feet. The end of the sluice is 3 feet above the deck, and extends 6 feet beyond the end of the boat. Quicksilver is used in the upper boxes. Seventy-five per cent of the gold is caught in the first 6 feet, and the balance in the next 8 feet. The tailings are frequently panned, but no gold found.

The description of this dredge, the cost of which should not exceed $25,000, laid down, is given in detail, as it is reported on good authority to have been successful in its operations. It presents a strong confirmation of the views expressed by Mr. Hutchins concerning the advisability of installing prospecting dredges for Alaska practice.

During the season of 1903 and 1904 a dredge of the dipper type was operated in Solomon River, near the one above described. The dredge was constructed originally with a stacker, but this system was abandoned and a separate hull was built on which a hopper and sluice were erected.

The dredge cost $33,000 on the ground, and weighs, barges and all, 400 tons. It is equipped with one yard dipper, which digs three-fourths cubic yard of material each time on an average. During 1904 the dredge was in operation fifty days of twenty hours each, and handled 21,000 cubic yards, or an average of 420 cubic yards a day.

The ground dredged is 9½ feet in average depth, and was all handled by the dipper. No permanent frost was found in the river bed, but the annual frost did not leave the ground until July 15, delaying the commencement of operations until that date. Thirty days at the end of the season were lost on account of the breaking of the main cable operating the dipper, and there was no chance to repair it.

The experiences with the ground in this dredging operation were especially interesting. It was found that in places the gravel in the river itself was frozen all summer. On July 20 there were 7 inches in depth of frost within 2 feet of the surface. This disappeared with the first heavy rain. The dipper cleaned the bed rock successfully where it was black schist. Where the bed rock was limestone the ground was drifted in the winter with much more satisfactory results than the dredging gave.

The dredge employed in the two shifts seven men, at $7 a day, and one man, at $9.50 a day, without board. In twenty hours 3 tons of coal, costing $26 per ton, were consumed.

The cost of dredging in 1904, including repairs to and maintenance of plant, was 50 cents per cubic yard. As in the case of the small dredge, the pay over a width of 200 feet, which was worked in places, was found to be spotted, varying from 30 cents to $12 to the cubic yard.

It is said that the clay in the gravel prevented the water from sluicing gold from the bottom of the dipper, and tests showed that a saving of 95 per cent was made. The sluice was paved with Hungarian riffles, and quicksilver was used.

The general opinion expressed by these operators was that dredges having a capacity of not to exceed 1,000 cubic yards daily will pay in Solomon and similar rivers of Seward Peninsula if operated by electricity generated from a central water-power plant. The employment of gravity-sluicing water was made a necessary condition. While this is possible, it should be borne in mind that the employment of gravity-sluicing water on a dredge entails numerous and largely untried difficulties.

Some of the special difficulties actually experienced in the operations above described were distance from any repair shop, sudden rises of the river after two days' storm that prevented persons from crossing and endangered the dredge, and repairs which were necessary at the beginning of the season, and which included chopping ice out of the inside of the hull.

The dredge shown on Pl. XXX, *B*, in Niukluk River, was operating only in prospecting work during the season of 1904. It has 5-foot buckets, open connected. Its rated capacity is 2,000 cubic yards in twenty-four hours, but the actual capacity can not be given. The dredge was fitted with a rubber belt conveyor for the stacker. This is said to give much trouble, as when it begins to freeze at night in September the belt continually slips. It was said that in previous operations the dredge in ten hours used 5 cords of spruce wood, costing $12 a cord, and handled 800 cubic yards of material. Eight men were employed at $7 a day, gross. In the Niukluk River banks the bed rock, which is schistose limestone, is 10½ feet below the surface, and is overlain by 1 foot of moss or willow roots, 2½ feet of sand or muck, and 7 feet of rounded river wash mixed with angular limestone fragments. Willows grow on the unfrozen ground and moss covers the solidly frozen ground. No regularity in the distribution of the two is apparent.

The success of dredging operations in the Niukluk appears to be extremely problematical. So far as the physical nature of the gravel goes, no difficulty should be experienced. Frozen areas, hard bed rock, uneven depth, and irregular distribution of the alluvial gold are, however, serious if not insurmountable obstacles to the success of a dredging operation which in a superficial examination of the ground may appear attractive.

## COST OF DREDGING.

It is manifestly impossible to give detailed statements of operating expenses in a work of this character, especially when dredging has been conducted in a desultory fashion and for a limited period. The costs given on page 38 include a certain amount each year for the amortization of the plant and the ground. Bearing this in mind, those having the slightest familiarity with the subject will probably admit that the figures, namely, 49 cents per cubic yard for unfrozen, 80 cents for frozen ground in interior Alaska, and 43 cents per cubic yard for Seward Peninsula, are not excessive.

For purposes of comparison there is given the following statement of the cost at Oroville, Cal., quoted in the Mining and Scientific Press, February 18, 1905, from the report of the Oroville Dredging and

Exploration Company to the stockholders for the year ending December 31, 1904, by F. W. Bradley, president. The figures given represent merely operating expenses, and do not include allowance for depreciation of the plants and of the ground. It should be remembered, however, that even adding these items of expense to the operating cost per cubic yard the number of yards handled per season is so large, as compared with that possible in northern operations, that the total California cost is not greatly increased. Eight cents per cubic yard is believed to be a safe estimate for the present total cost of gold dredging at Oroville.

The table of lost time in the following quotation is especially interesting as showing that even under the favorable California conditions the actual dredging time of a presumably representative dredge amounted to only 67.2 per cent of the whole.

### DREDGING AT OROVILLE, CAL.

Our No. 1, or $45,000 dredge, made the following record:

| | |
|---|---|
| Cubic yards dredged | 493,150 |
| Gold yield per cubic yard, $0.1232; operating costs, $0.0562; operating profit, $0.067, or | $32,905.79 |
| Improvement, taxes, insurance, legal, and all other working expenses (except construction of No. 2 dredge) | 7,166.88 |
| Surplus | 25,738.91 |

Our No. 2, or $75,000 dredge, should be ready to begin work next month (January, 1905). Figuring on the data now available, the two dredges should make the following average annual record:

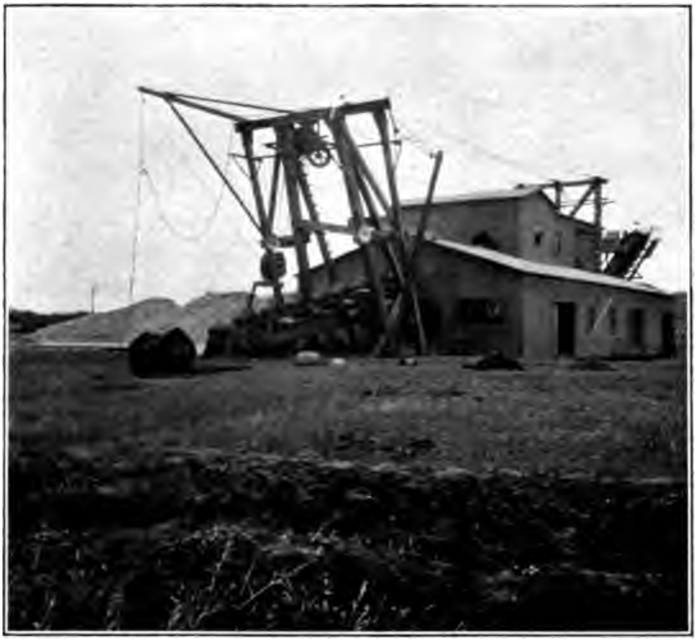
| | Dredged. | Operating cost. | Probable yield. | Surplus. |
|---|---|---|---|---|
| | *Cubic yards.* | *Per cu. yd.* | *Per cu. yd.* | |
| Dredge No. 1 (first cost $45,000) | 520,000 | $0.0530 | | |
| Dredge No. 2 (first cost $75,000) | 780,000 | .0500 | | |
| Total or average | 1,300,000 | .0512 | | |
| Improvements, taxes, and all other working expenses | | .0088 | | |
| Total | | .0600 | $0.095 | $0.035 |

Or a yearly surplus of $45,500. Our prospect information indicates that after rejecting that portion of our land of doubtful value the property should have a life of from thirteen to fourteen years at the above estimated rate of dredging.

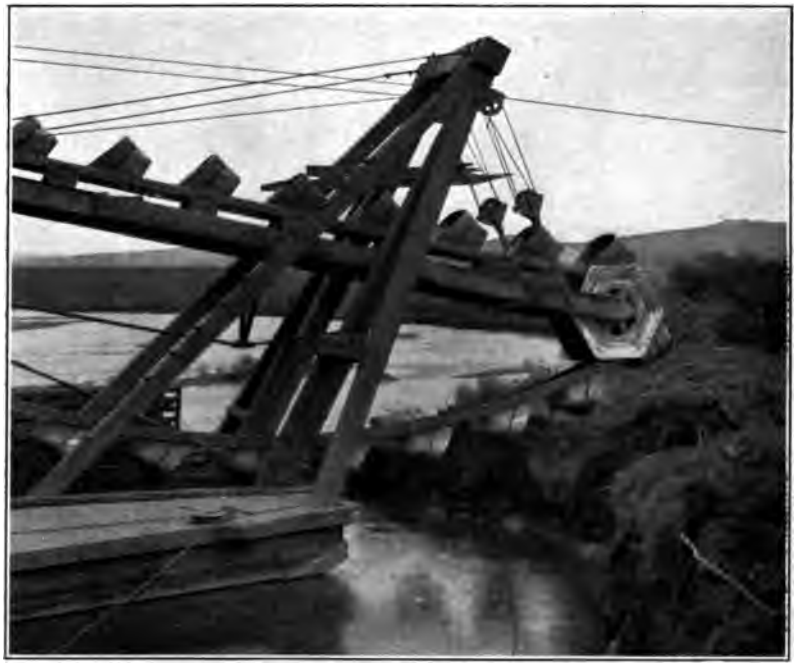
C. H. Munro, superintendent, makes a detailed report of the company's operations, from which the following is taken:

*Extraction.*—The prospect value of the area dredged, based on the average value of the holes in and near the dredged area, was 16.64 cents. The extraction table herewith attached, which is computed on these figures, makes the extraction 74 per cent of the prospect value.

The extraction amounted to 76.37 per cent of the prospect value, based on the *proportional* or fractional areas as influenced by each prospect hole.



*A*. DREDGE OF NEW ZEALAND TYPE AT OROVILLE, CAL., LEGGETT NO. 3.



*B*. BUCKETS, DREDGE AT COUNCIL.

It would therefore be fair to call the extraction 75.18 per cent of the prospect value, which is the average of the two methods of computation above referred to.

During the year just ended, Biggs No. 1 dredged 11.85 acres, handling 493,150 cubic yards, which yielded a gross bullion return of $60,738.52, or an average yield of 12.32 cents per cubic yard, as compared with a gross bullion return of $36,525.25, or an average yield of 8.45 cents per cubic yard for 1903.

*Costs.*—The operating cost has been 5.62 cents per cubic yard, and the total cost, including all improvements, repairs, taxes, insurance, legal, office, and other expenses (excepting Biggs No. 2 construction account), has amounted to 7.06 cents per cubic yard.

Comparing the capacity of the dredge and the cost of operation for 1904 with 1903, the capacity has increased from 39,551 cubic yards per month to 41,096 cubic yards per month, while the cost of operation has decreased from 6.48 cents per cubic yard to 5.62 cents per cubic yard, and the total expense from 7.71 cents to 7.06 cents per cubic yard.

The bullion yield has increased from 8.45 cents per cubic yard to 12.32 cents per cubic yard.

*Repairs.*—Aside from the ordinary repairs on the dredge, a new revolving screen was installed at a cost of $1,600, and the 1-horsepower motor used for priming the water pumps and cleaning up was replaced by a 3-horsepower motor at a cost of $127.55.

*Flume repairs.*—During the floods last February 500 feet of the flume and trestle were washed away. The rebuilding of the flume and trestle and repairs at the pump station cost $357.40.

*Improvements.*—The blacksmith shop equipment has been enlarged at a cost of $547.45 by installing a 5-horsepower motor, with transformer, a forge, blower, emery wheels, grindstone, and small drill press.

*Biggs No. 2 construction.*—The hull of Biggs No. 2 is completed and the dredge was floated the first of the year. The contractors are now installing the machinery, and the dredge will probably be ready for operation about the middle of February, 1905.

*Bullion table.*

| Month. | Cubic yards bank measurement dredged. | Bullion yield. | Operating expenses. | Yield per cubic yard. | Operating cost per cubic yard. | Operating profit per cubic yard. |
|---|---|---|---|---|---|---|
| | | | | *Cents.* | *Cents.* | *Cents.* |
| January | 20,340 | $3,120.87 | $2,677.00 | 15.38 | 13.16 | 2.22 |
| February | 32,000 | 3,408.59 | 2,205.07 | 10.65 | 6.89 | 3.76 |
| March | 41,160 | 7,732.62 | 2,418.94 | 18.78 | 5.88 | 12.90 |
| April | 50,760 | 9,600.10 | 1,854.69 | 18.91 | 3.65 | 15.26 |
| May | 47,700 | 8,072.17 | 2,233.41 | 16.94 | 4.68 | 12.26 |
| June | 40,750 | 3,704.36 | 2,013.97 | 9.09 | 4.94 | 4.15 |
| July | 44,380 | 4,271.03 | 2,272.36 | 9.62 | 5.12 | 4.50 |
| August | 39,300 | 4,273.31 | 2,471.91 | 10.87 | 6.29 | 4.58 |
| September | 41,600 | 4,265.74 | 2,392.09 | 10.25 | 5.75 | 4.50 |
| October | 48,460 | 4,645.08 | 2,620.83 | 9.58 | 5.40 | 4.18 |
| November | 40,300 | 3,374.92 | 2,340.96 | 8.37 | 5.81 | 2.56 |
| December | 46,400 | 4,269.73 | 2,331.31 | 9.20 | 5.02 | 4.18 |
| Total or average | 493,150 | 60,738.52 | 27,832.53 | 12.32 | 5.62 | 6.70 |

*Extraction table.*

| Month. | Depth in feet. | Dredged. | | Number prospect holes in and near dredged area. | Prospect value of ground dredged. | Prospect value of gold recovered. | Total gold recovered in cents per cubic yard. | Operating cost in cents per cubic yard. | Character of ground dredged. |
|---|---|---|---|---|---|---|---|---|---|
| | | Acres. | Cubic yards—bank measurement. | | | | | | |
| | | | | | *Cts.* | *Per cent.* | | | |
| January | 32.1 | 0.40 | 20,340 | 2 | 15.35 | 100.20 | 15.38 | 13.16 | Coarse gravel to surface; dredging across slough. |
| February | 22.0 | .90 | 32,000 | 2 | 15.35 | 69.57 | 10.65 | 6.89 | 10 feet top soil; 2 feet of gravel on bed rock; hard digging. |
| March | 28.7 | .89 | 41,160 | 2 | 37.70 | 50.00 | 18.78 | 5.88 | 10 feet soil; gravel coarse, and hard digging. |
| April | 27.0 | 1.15 | 50,760 | 3 | 37.70 | 50.00 | 18.91 | 3.65 | 12 feet top soil; 15 feet coarse gravel; 3 feet on bed rock; hard digging. |
| May | 27.0 | 1.09 | 47,700 | 3 | 16.25 | 104.24 | 16.94 | 4.68 | 12 feet top soil; gravel coarse. |
| June | 21.4 | 1.18 | 40,750 | 2 | 10.35 | 87.96 | 9.09 | 4.94 | 10 feet soil; gravel firm, and sand on north side of cut. |
| July | 22.0 | 1.25 | 44,380 | 2 | 11.45 | 84.02 | 9.62 | 5.12 | 4 feet top soil; coarse gravel on west side cut; sand streak on east side. |
| August | 26.0 | .93 | 39,300 | 2 | 11.45 | 94.93 | 10.87 | 6.29 | 6 feet top soil; sand streak on west side of cut; all fine gravel; easy digging. |
| September | 24.2 | 1.06 | 41,600 | 2 | 4.40 | 230.00 | 10.25 | 5.75 | 6 feet top soil; sand on west side of cut; all fine gravel; easy digging. |
| October | 23.8 | 1.27 | 48,460 | 3 | 5.25 | 182.48 | 9.58 | 5.40 | 6 feet top soil; sand in center of cut; remainder fine gravel; easy digging. |
| November | 31.0 | .81 | 40,300 | 3 | 17.23 | 48.58 | 8.37 | 5.81 | 7 feet top soil; east side cut in sand; remainder of cut fine, and coarse gravel. |
| December | 28.8 | .92 | 46,400 | 3 | 17.23 | 53.40 | 9.20 | 5.02 | 7 feet top soil; coarse and fine gravel full width of cut. |
| Total or average | 25.8 | 11.85 | 493,150 | 29 | 16.64 | 74.00 | 12.32 | 5.62 | |

The figures for the several items of expense in the following tables varied from month to month throughout the year, and the averages for the entire year only are given:

*Operating and all other costs.*

| | Total for year. | Total expense per cubic yard. |
|---|---|---|
| | | *Cents.* |
| Operating expense Biggs No. 1 and superintendence | $27,832.53 | 5.62 |
| General expense, Oroville and San Francisco | 1,788.62 | .36 |
| General plant | 1,262.78 | .26 |
| Bullion expense | 251.34 | .05 |
| Taxes | 1,942.66 | .39 |
| Warehouse | 401.18 | .08 |
| Insurance | 450.00 | .09 |
| Legal | 1,070.50 | .21 |
| Total | 34,999.41 | 7.06 |

*Details of operating costs.*

| | Total for year. | Per cent of total. | Cost per cubic yard. |
|---|---|---|---|
| Labor: | | *Per cent.* | *Cents.* |
| Operative | $6,435.20 | 22.9 | 1.290 |
| Repair | 4,690.41 | 16.8 | .943 |
| Superintendence | 1,875.00 | 6.8 | .380 |
| Power: | | | |
| Dredge | 4,902.81 | 17.7 | 1.000 |
| Pumps | 68.20 | .3 | .014 |
| Hardware: | | | |
| Supplies, tools, oils, etc | 1,109.77 | 4.0 | .220 |
| Repair parts, etc | 6,768.23 | 24.4 | 1.370 |
| Freight, express, and hauling | 504.69 | 1.8 | .100 |
| Steel cables | 336.03 | 1.2 | .070 |
| Lumber | 28.06 | .1 | .006 |
| Electric supplies and work | 285.67 | 1.0 | .060 |
| Clearing ground | 794.40 | 2.9 | .160 |
| Sundry expenses | 34.06 | .1 | .007 |
| Total | 27,832.53 | 100.0 | 5.620 |

*Repair table showing cost of labor and material on different parts of the dredge.*

| | Total for year. | Grand total. |
|---|---|---|
| Ladder: | | |
| Labor | $492.63 | |
| Material | 596.43 | $1,089.06 |
| Bucket line: | | |
| Labor | 1,328.07 | |
| Material | 3,237.35 | 4,565.42 |
| Stacker: | | |
| Labor | 894.66 | |
| Material | 1,125.04 | 2,019.70 |
| Winches: | | |
| Labor | 138.73 | |
| Material | 84.05 | 222.78 |
| Screen: | | |
| Labor | 772.42 | |
| Material | 1,498.57 | 2,220.99 |
| Pumps: | | |
| Labor | 125.50 | |
| Material | 16.80 | 142.30 |
| Power: | | |
| Labor | 311.80 | |
| Material | 541.59 | 853.39 |
| Lines: | | |
| Labor | 333.43 | |
| Material | 361.48 | 694.91 |
| Gold tables and clean-ups: | | |
| Labor | 152.03 | |
| Material | 74.71 | 226.74 |
| Contingent repairs: | | |
| Labor | 800.56 | |
| Material | 1,307.59 | 2,108.15 |

*Table of lost time.*

| | | |
|---|---|---|
| Ladder | per cent.. | 10.7 |
| Bucket line | do... | 20.5 |
| Stacker | do... | 10.7 |
| Winches | do... | 3.5 |
| Screen | do... | 9.9 |
| Pumps | do... | 2.6 |
| Power | do... | 7.9 |
| Lines | do... | 9.0 |
| Clean-up | do... | 3.5 |
| Other causes | do... | 21.7 |

| | |
|---|---|
| Total lost time ........ per cent.. | 32.8 |
| Dredging time ........ do... | 67.2 |
| Average number of cubic yards dredged per day of running time | 2,005 |
| Actual average number of cubic yards dredged per day of twenty-four hours.. | 1,347 |

*Summary of costs in cents per cubic yard, 1904.*

| | |
|---|---|
| Dredge crew and power | 2.69 |
| Repair labor | .94 |
| Repair supplies | 1.61 |
| Superintendence | .38 |
| Oroville and San Francisco general expense | .36 |
| Taxes and insurance | .48 |
| Bullion expense | .05 |
| Grand total, all costs | 6.51 |

For three $45,000 dredges these items would have been as follows:

| | |
|---|---|
| Dredge crew and power | 2.69 |
| Repair labor | .66 |
| Repair supplies | 1.61 |
| Superintendence | .17 |
| Oroville and San Francisco general expense | .12 |
| Taxes and insurance | .27 |
| Bullion expense | .05 |
| Grand total, all costs | 5.57 |

Mr. J. P. Hutchins, of San Francisco, who has had long experience in the dredging fields of Oroville and has recently worked in the Klondike, has courteously furnished for this report an account of the present status of the industry, with suggestions regarding the forms of dredges most applicable to Alaska conditions.

## NOTES ON DREDGING.

By J. P. Hutchins.

Dredge mining of placer gold has been one of the most attractive fields for investments of capital in recent years. Prospecting methods in use to determine the values of the areas thought to be available for profitable exploitation by dredging are not complicated. Until recently well-drilling machines have been used almost exclusively in this work, and in many cases the results of such sampling have led to large investments of capital in dredging ground and dredging machinery. These drilling machines have much to recommend them. They are cheap and mobile, can be operated on floating scows to sample river, lake, or sea bottoms (see Pl. III, *A*, p. 40), or used to sample a deposit where a large volume of water near the surface would make shaft sinking very costly. Numerous holes can be drilled in a short time and at a comparatively small cost. In many cases shafts have been sunk with the drill holes for centers, and the results of the two methods of prospecting checked well.

It was first held that the results of dredging could be relied upon to equal about 50 per cent of what drill prospecting would indicate the values to be. The disparities were in part accounted for by attributing them to spilling, leakage, faulty discharge of buckets, incomplete excavation by apparatus, incomplete disintegration during washing in screening, and to losses in the gold-saving apparatus. It seemed, however, as if these losses should be only about 10 per cent, and the other 40 per cent was explained by the well-known general statement that "prospecting results are always high."

Up to the time that considerable areas previously prospected with numerous drill holes were dredged, the results of drill prospecting were believed to be absolutely reliable, making the allowance, however, of about 50 per cent for values to be recovered by the dredge. After these areas had been worked, however, the dredge managers found much to surprise them, for where ground had prospected well it had dredged poorly, and vice versa. It is asserted that in these instances both the drilling and dredging were so conducted that no great reliability could be attached to the results, but this explanation is far from satisfactory.

In the use of the drilling machine to sample alluvion, as in all other prospecting, unusual care must be exercised to obtain results that shall give a correct idea of the values and of the other characteristics of the ground under examination. It is unquestionably true that careless sampling has been the cause of many failures in dredging operations. The seeming simplicity of this prospecting method has, as in other apparently simple determinations, resulted in crude manipulation and misleading results. In some instances there has been a misleading affectation of accuracy in tabulating results of drill prospecting. Careless location and distribution of drill holes have given faulty results. Nevertheless, the percussion drilling machine is peerless for testing most areas that may be available for exploitation by the dredging method.

The drilling machines are used for several purposes. It is necessary to determine whether the material is too hard to make dredging profitable, and for this purpose drilling machines are not entirely satisfactory. In several areas prospected by drilling machines and thought suitable for dredges the material proved to be so hard that a machine different from the one designed was required for successful exploitation. If the prospecting had been done with more care this unfavorable condition would have been learned.

Reliable results call for the determination of the presence and distribution in the vertical section of any considerable content of tenacious clay and large bowlders, and here again the drill does not yield entirely satisfactory results. The depth of material must be ascertained and, where the bed rock is a so-called "false," the greatest care is necessary in order to determine its position. The character, hardness, and roughness of the bed rock must be learned, and the drill unfortunately still leaves much to be desired in connection with this phase of the investigation. In spite of these drawbacks the percussion drilling machine has no superior as a preliminary means of prospecting alluvions thought to be suitable for working by the dredging method, and this is particularly true where large volumes of water may be encountered near the surface. The sinking of shafts is to be preferred, however, where not too costly.

In several projects, where a large expenditure of money is in contemplation, so-called prospecting dredges have been used. These machines are small but complete dredges, and perform, on a small scale, the functions of the ordinary type. Their design is such that they may be readily moved, for they are equipped with lightly constructed machinery and hulls and are of light draft. If, as is frequently the case, the area to be prospected is traversed by a river or sloughs, such a light dredge is easily moved from place to place. Where there are no waterways, a still smaller dredge, to be moved on rollers or skids, can be employed.

Though the cost of installation and operation of such prospecting dredges is considerably more than that of drilling machines, the reliability of the results will often justify the additional expenditures, particularly where large areas are being prospected. By their use the values per cubic yard, character of material, presence of large bowlders and clay, and depth and character of bed rock can be accurately ascertained. There is an additional advantage of great importance in the fact that the cost of operating a large dredge can at the same time be closely determined.

The number of gold dredges in successful operation in the United States has greatly increased during the past decade, and especially in the last six years. The first successful one was operated at Bannock, in Montana, and was of the so-called "double-lift" type. The material was excavated by a chain of buckets and discharged from an elevation of about 15 feet above the deck into a trommel, which had perforations about 5 inches in diameter. Stones of larger diameter were discharged over the side of the dredge; the "fines" were elevated by a 12-inch centrifrugal dredging pump to a sluice and discharged about 100 feet astern. This type of dredge is still advocated by some managers, and unquestionably has a number of advantages over other types. It is especially good for handling material which has a clay content or which is so tenacious that gold is lost by incomplete disintegration and insufficient washing in the screens and sluices of the other types of gold dredges. The passage of the fines through the centrifugal pump of the double-lift dredge tends to disintegrate the material thoroughly and to completely liberate any particles of gold held in clay or other tenacious substances.

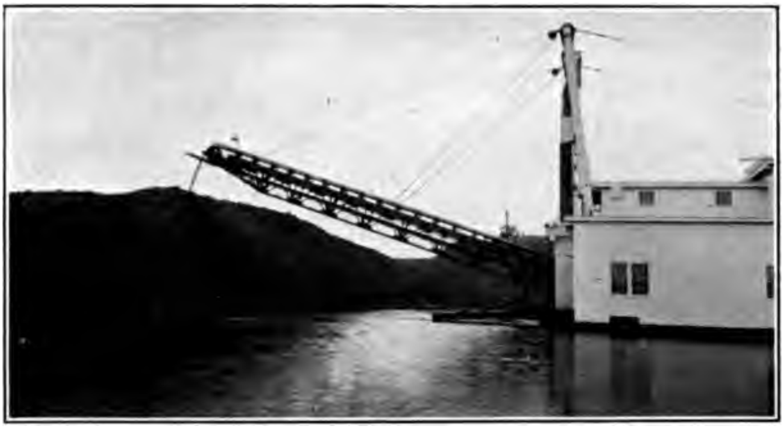
The particular problem at Bannock was to evolve a dredging pump to stand the wear and tear of handling the fines of about 2,000 cubic yards every twenty-four hours. The first pump was worn out after seventy hours' use, but another of excellent construction, designed by the dredge manager, was installed and used for several years. After a seven months' run, when the fines of over 400,000 cubic yards had passed through it, it was estimated that the costs of repairs and renewals of the pump had been only about $150.

With a view of eliminating the dredging pump the so-called single-lift dredge was designed by Montana men. This type discharges the material about 28 feet above the deck into a trommel with perforations 5 inches in diameter. As in the "double-lift" dredge, the coarse material was deposited over the side, while the fines dropped into a sluice and were discharged about 100 feet astern. The greater height of discharge of buckets made a secondary lift of the fines unnecessary. This form of construction introduced a new set of problems, as the higher position of the upper tumbler necessitated longer ladders, more and heavier buckets in a longer bucket chain, larger hulls, and heavier construction throughout. Some of the first "single-lift" dredges were very unstable because of the high position of center of gravity. This type of dredge is now in more general use than the "double-lift" machines.

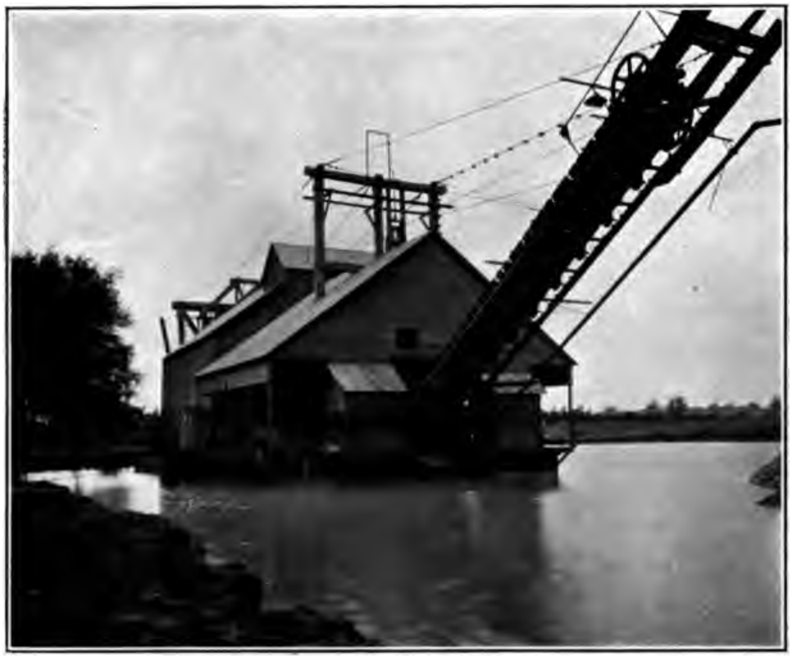
The dredge which had been used so successfully in New Zealand was not introduced into the United States until after the extensive operation of these two types. The first New Zealand dredge was installed in the Oroville district of California, and after a few modifications to suit local conditions was and is still operated successfully. This was followed by the building of a number of similar dredges in California, Colorado, Idaho, Oregon, Montana, and elsewhere (see Pls. XXX, *A*, and XXXI, *B*).

These dredges were essentially different in design and manipulation from those of the double and single lift types. While the bucket of the New Zealand type was evolved entirely for gold-dredging purposes, the bucket previously used in this country was but a slight modification of the one used for harbor and canal dredging. In both types, but more notably in the bucket evolved in harbor and canal dredging, the first made were deficient in the strength of the digging apparatus.

The New Zealand type of gold dredge discharges the material about 16 feet above the deck into a trommel, the largest perforations of which are about one-half inch in diameter. A stacker (see Pl. XXXI, *B*) made of a chain of steel buckets elevates all material not passing through perforations of trommel and discharges it astern of the dredge at a height sufficient to allow for the expansion of the tailings. The fines pass over the gold-saving devices and are dis-



*A* DIGGING ON SPUD.



*B.* STACKER, NEW ZEALAND TYPE OF DREDGE, CALIFORNIA NO. 1.

charged about 10 to 15 feet astern of the dredge. On the double and single lift dredges no stackers were necessary, because the larger perforations of the trommels allowed the comparatively small percentage of stones to be discharged over the side without shoaling the water too much.

The gold-saving apparatus introduced with the New Zealand type was also essentially different. The hydraulic mining undercurrent, which is a wide sluice with steep grade, and which spreads the material over a broad surface in a shallow sheet after all stones of diameter greater than 2 inches have been removed by a grizzly, was evolved in California but had been successfully used in New Zealand in hydraulic operations. A very slightly modified form of this device was used on a number of the dredges, with the difference that cocoa matting instead of riffles charged with quicksilver was used to catch the gold (see Pl. XXIX, *A*, p. 162).

The double and single lift gold dredges depended principally, and in some cases entirely, on a comparatively narrow sluice with riffles charged with quicksilver. The material passed over the riffles in a deep sheet, a condition not conducive to a high percentage of saving, where the gold is in the form of very finely divided particles.

There is a marked difference in the manipulation of the several types of dredges. The New Zealand boat in digging is held to the face of the bank by a headline run out over the bow to an anchorage, and the buckets are lowered in a vertical plane, the material being caved by undermining at the bottom. The other types are held to the face by a pivotal stern spud (see Pl. XXXI, *A*), the buckets being side fed horizontally through an arc of 120°. The cut is begun at the surface and the digging ladder lowered about 1 foot at the completion of each arc, no attempt being made to undermine or cave the material excavated.

Several shovel dredges were built at Oroville at about the time the New Zealand type was installed. On these the hydraulic-mining undercurrent, almost the same as that of the hydraulic miner in California, and a shaking screen, similar in action to the rocker of the early-day California miner, are used. Belt stackers were also installed. The shovel dredges proved costly of operation and have, with few exceptions, been discarded.

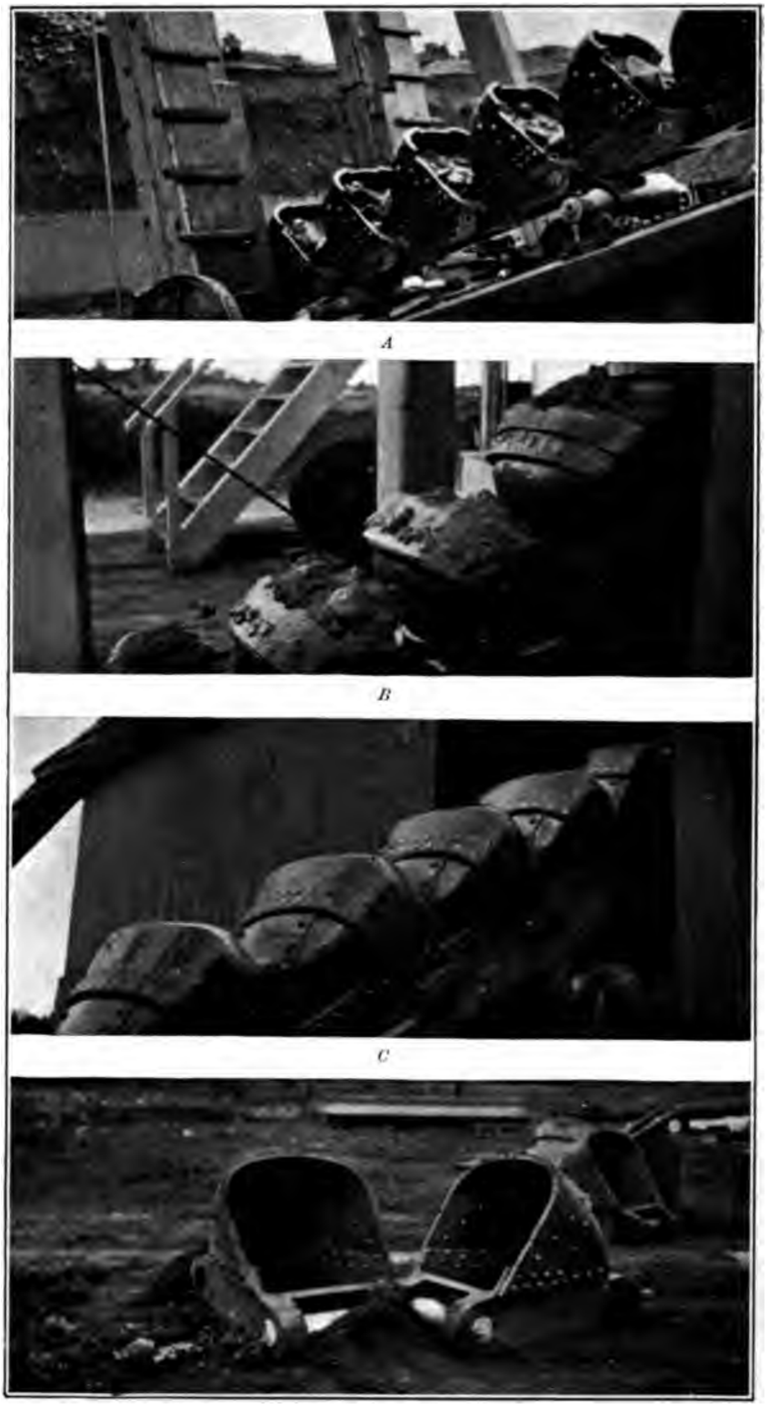
Next in order of development came a dredge with the bucket evolved in canal and harbor dredging. The machine was very much strengthened, however, and was provided with shaking screen, belt stacker, and hydraulic-mining undercurrents, called "gold-saving tables." In this type is found a combination of the better features of the double and single lift and shovel gold dredges. Small buckets were first used, but the tendency had been toward increased capacity. This dredge was an improvement, but it had a fault common to all the dredges at that time and still often found—a lack of coordination of capacity in the four processes of which gold dredging consists, namely, excavating, screening, sluicing, and tailing.

The next improvement was the building of a dredge in which coordination of capacities received more attention, for in this the screens, tables, stacker, and pumps were enlarged. It was designed to work either on spud or headline, with a slightly higher discharge of material to permit of a greater elevation of coarse tailings and a stacking of them at a distance from the stern. The pivotal spud was placed inboard so as to make the digging and the tailing arcs of more equal lengths when working on a spud. The difficulties encountered when these arcs were of very unequal length were thus eliminated. The disposition of the tailings had formerly been a continual source of difficulty and expense, but this improved dredge was so designed as to allow both working on a headline and side feeding through an arc of long radius. This permitted the working of a wide cut and the disposition of tailings without covering adjacent virgin ground, for previously much untouched territory had been wasted by being covered with tailings. Impact riffles charged with mercury, i. e., those where the fines are dropped to effect amalgamation, and larger gold-saving areas were also introduced.

There have been no radical changes in the design or construction of gold-dredging machinery of late, and efforts have been directed toward the strengthening and bettering of machinery and hull and the lowering of operating expenses.

The excavating apparatus (see Pl. XXXII; Pl. XXVIII, *B*, p. 160, and Pl. XXX, *B*, p. 168) used in gold dredging has been very much improved since the first installations. Bucket forms have been modified so as to prevent losses by spilling. This is accomplished by reducing the angle made by the bucket lip with the line through the centers of the two bucket pins. With the earlier design a considerable loss by spilling, except when buckets were working at maximum depth, was noticed. In the earlier design the bucket was long, narrow, and deep, and a clean discharge was not effected, especially when working in sticky or sandy material. Recent design provides a short, wide, and shallow bucket which discharges more perfectly. The use of jets of water to wash any adhering material from both inside and outside of buckets into the hopper and save-alls is now common and accomplishes a considerable increase in the gold saving. The capacity of buckets has been largely increased, but the economical limit has not been reached nor will it be reached in the near future.

With the increase of size of buckets there has not been a proportionately greater strength, but this defect is being gradually remedied. The use of various alloys of steel in bucket parts, as well as in other



BUCKETS.

*A*, California No. 3; *B*, El Oro; *C*, Indiana; *D*, Lava Beds No. 2.

parts of the dredging machinery where there is severe wear and tear, has been in most cases successful.

The design of the upper and lower tumblers has been modified and all parts subject to wear are now easily replaceable. In earlier installations the tumblers were of such design and construction that the wearing out of one part sometimes necessitated the loss of the whole. Some trouble has been caused by the working loose of the "wearing plates." Tumblers with more numerous faces are now also used. In early practice the square upper tumbler was supposed to be superior to the one with more faces, it being assumed that the square tumbler was more capable of firmly holding the buckets and preventing the danger of its revolving without moving the bucket chain. The hexagonal upper tumbler is now in successful operation, and it is safe to predict that heptagonal and possibly octagonal upper tumblers will be used, particularly on the smaller dredges. The objection that they will not hold is not good. Even if true, an arrangement making the relation of bucket to tumbler that of chain to sprocket would solve the difficulty. The use of the heptagonal and octagonal upper tumblers introduces a new set of problems, involving larger tumblers and shafting, different arrangement of chute to hopper, and modified forms of save-alls. These modifications are necessary because of the resulting longer period of discharge of the bucket. It is probable, however, that saving in wear of the bucket pins and bushings due to lesser deflection in turning tumblers of more numerous faces will compensate for the extra expense involved in construction.

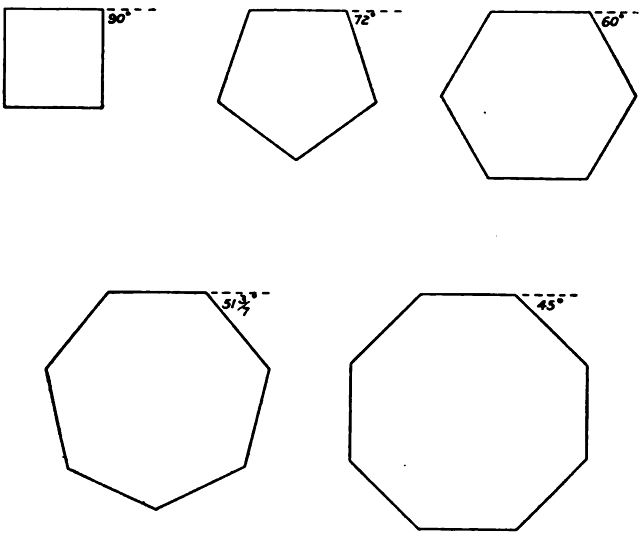


FIG. 34.—Angles for upper tumbler.

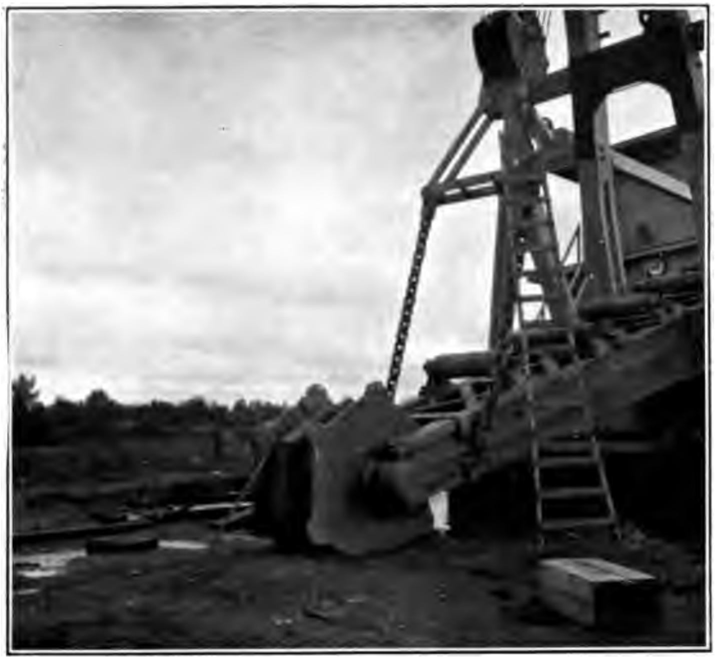
Fig. 34, above, shows the relative size and different deflection angles made by the buckets in turning the square, pentagonal, hexagonal, heptagonal, and octagonal tumblers and per cent of saving of wear over the square type. The wear and tear of bucket pins and bushings is almost entirely the result of the deflection of the buckets in rounding the tumblers. A slight amount of wear is occasioned by deflections back of the lower tumbler as the buckets come in contact with the material. These last deflections, however, would be prevented by rollers such as are suggested in a paragraph on page 184 discussing the respective merits of the spud and headline.

If, instead of square upper and lower tumblers, pentagonal, hexagonal, heptagonal, or octagonal tumblers are used the saving of wear of bucket pins and bushings would be approximately 40, 66, 86, and 100 per cent, respectively.

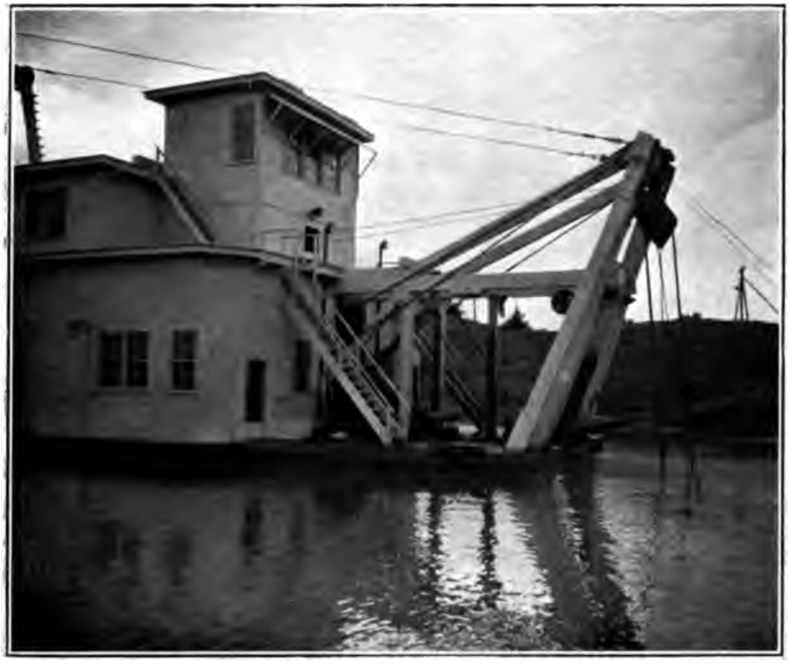
There is but one serious objection to the use of a lower tumbler of more numerous faces, i. e., greater size and weight. The great saving in wear of the bucket pins and bushings will, when fully appreciated, result in the use of both upper and lower tumblers of more numerous faces. (See Pl. XXXIII, *A*.)

One dredge manager of much experience has suggested the use of a round lower tumbler of large diameter with buckets of short pitch, claiming that buckets seldom lie flat on tumbler faces when excavating, but frequently ride the corners between the faces. This difficulty has been observed by the writer on many occasions. A too rapid feeding of buckets resulted in considerable spilling of material upon the buckets about to round the lower tumbler. The spilled material lodged between the buckets and the tumbler faces and destroyed, temporarily, their proper relation. The use of the round lower tumbler would have the additional advantage of preventing surging (see p. 184).

Until recently it has not been generally regarded as good practice to lower the digging ladder to more than 45° from the horizontal because of supposed disadvantages in such manipulation. Recent installations provide an idler at the rear end of the well to prevent the buckets from dragging on the chutes and the save-all gratings as the ladder is lowered more than 45° from horizontal. This device permits greater working depths with a given length of ladder and makes deeper ground available for the dredge. Pl. XXXIII, *B*, illustrates diggings at a low angle. There is a marked diversity of opinion among managers as to the relative merits of the trommel and shaking screen. The trommel advocates have claimed better mechanism, cheaper maintenance, more efficient screening when material is at all tenacious, and economy in power. The advocates of the shaking screen have claimed larger capacity with given weight, higher percentage of gold saving because of better opportunities for amalgamation by impact, and easier and better distribution of material over the gold-saving surfaces.



*A*. LOWER TUMBLER, CALIFORNIA NO. 3.



*B*. DIGGING LADDER AT LOW ANGLE, OROVILLE, CAL

The trommel (see Pl. XXIX, *A*, p. 162) is a better mechanism, requires less power for operation, and gives much less trouble in maintenance than the shaking screen and does cleaner screening, particularly when the material is at all indurated or tenacious. Trommels have almost without exception been too small, and all late installations are larger. Amalgamation by impact is now accomplished by a special arrangement of the gold-saving apparatus, and distribution of water is no longer a difficult problem on dredges where trommels are used. One of the most recent installations by a company which has extensively employed both devices includes a trommel. At present in most cases the trommel is in high favor among Oroville operators.

Where there is an overburden of any considerable depth the use of a device to blank the trommel or screen will be a distinct improvement. By its use all barren material can be stacked well back of dredge and usually at a more rapid rate than if handled in the ordinary way. Overburdens, which it is always necessary to dispose of as quickly and cheaply as possible, usually consist of a fine loam or sand which can be run but slowly over the gold-saving devices because of clogging when handled rapidly. A blanking device was used for a time at Oroville and worked well, but, because of bad design, required too much time in blanking and unblanking the screen. Its use, however, allowed the beginning of dredging operations, on interior ground, without the use of a sand pump and the immediate excavation to a depth of 30 feet, with a consequent minimum wasting of dredging ground. The use of such a device involves a large discharge of material from screen or trommel to the stacker where a troughed belt is used. The incline of the stacker must not be so steep as to permit the material, which is often a thin mud, to run back to the end.

The use of a device to blank the hopper which shunts the material into the well has been tried, but has not been entirely successful. In this case also, bad design rather than bad principle was responsible for failure. The disposition of an overburden by this method is to be heartily recommended, especially where dredges work on headlines, as they are thus enabled to distribute the barren material across most of the width of the cut.

The gold-saving apparatus, after many experimental modifications, including the introduction of various magnetic and intricate devices, has returned to the hydraulic-undercurrent type with the mercury riffle. For saving clean gold the mercury riffle is probably best. When, however, for any reason there is imperfect amalgamation, the use of some form of woven material to enmesh particles of rusty gold is good practice.

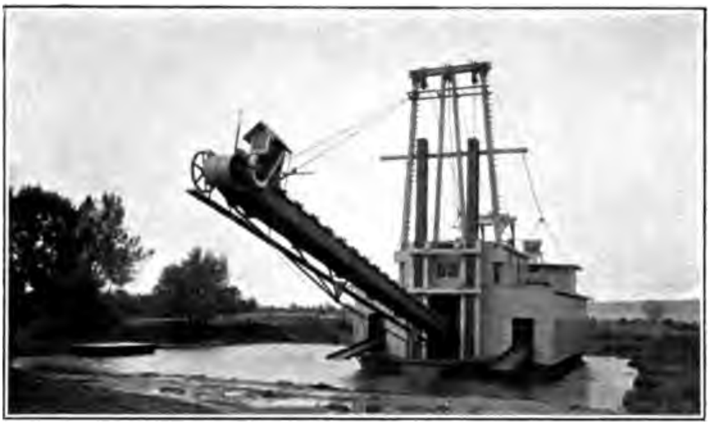
The increase of the area of gold-saving surface has met with good results. Steeper grades are now used for gold-saving tables than formerly, and arrangements are such as to allow a better distribution of sluicing water to various parts of the gold-saving areas. Gold-saving surfaces are now generally arranged in a number of drops, and the material impinges upon riffles charged with mercury and is amalgamated by impact.

Much thought has been recently given to the perfecting of save-alls, and a surprisingly large amount of gold, sometimes 10 per cent of all saved, is recovered in this way. The design of save-alls in the earlier practice was somewhat neglected, but in the general refinement of working methods a considerable improvement is being effected. A device for keeping under lock and key parts of the gold-saving apparatus where the largest proportion of gold is caught has been introduced.

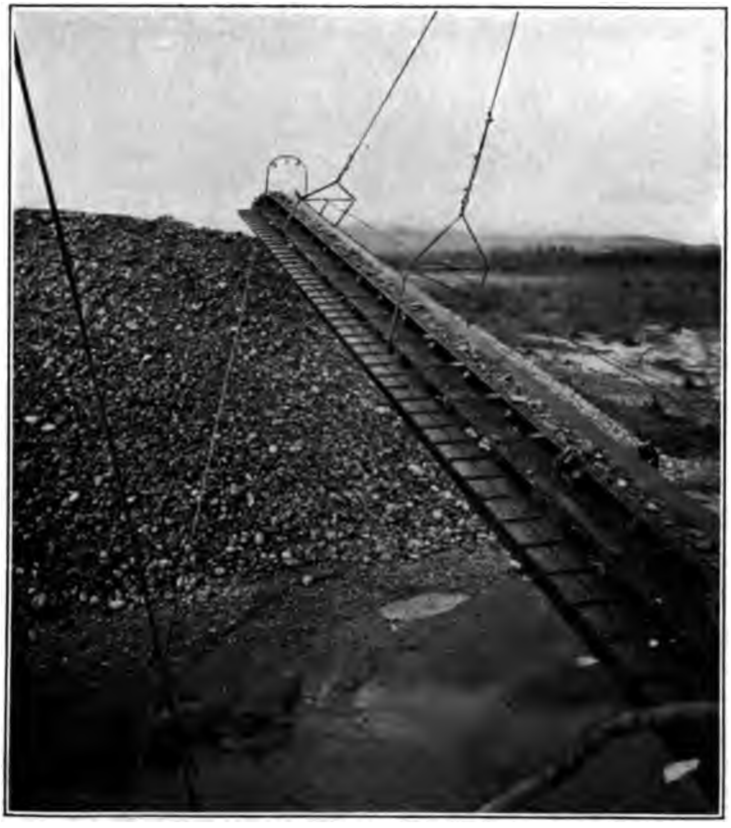
The troughed belt stacker seems to be generally regarded as superior to the bucket stacker (see Pl. XXXIV, *B*) because of greater economy of operation and less weight for a given length. Stacker belts are a source of large expenditure because of the rapid wear due to the insufficient thickness and inferior quality of the material which covers the woven part of belt and which gives the belt its wearing qualities.

Nearly all the recently installed stackers have the driving pulley on the outboard end, and electric dredges have the driving motor, as well, on the outboard end. (See Pl. XXXIV, *A*.) This arrangement has eliminated a number of troublesome features. If the stacker is driven by rope or sprocket chain from a motor on the dredge, there is much trouble due to wear, slipping, and stretching of ropes and of sprocket chains. If the driving pulley is at the inboard end of the stacker, the load is pushed uphill on the slack side of the belt, and there is much trouble from slipping and stalling. A belt stacker driven by the inboard pulley requires more power than one operated by the outboard pulley.

At one time it was generally thought that when coarse gold occurred in the material dredged, either the double or single lift type of dredge must be used. Stacker dredges are now used and have at the lower end of the trommel perforations large enough to pass material up to 1½ inches in diameter. This material and water run through a sluice where gold is saved and are then deposited in the stacker buckets. With such a device under ordinary conditions a troughed belt stacker could not be used, as it could not handle the water. Any device that would remove water and deposit pebbles on the belt without too many complications would allow the use of the belt stacker. (See Pl. XXXV, *A*, p. 184.) It would not be expedient to use a double or single lift dredge where ground is deep. To effect disposition of tailings while operating the double and single lift dredges it is necessary to keep the pond nearly full and to use very long digging ladders.



*A*. UNCOMPLETED BELT STACKER, OROVILLE, CAL.



*B*. TROUGHED BELT STACKER, OROVILLE CAL.

Stackers and tail sluices are of greater length than formerly. In early installations, without a single exception, stackers and tail sluices were too short, and difficulties due to improper disposition of tailings were of constant occurrence. Considerable areas of rich dredging ground, particularly when deep, have not been thoroughly worked, and much virgin ground has been covered by tailings to such a degree as to make it unworkable. The short stacker does not allow the coarse tailings to be discharged at a sufficient distance from the stern and at the proper height. There is consequently lack of room for fine tailings. In consequence the sand pump was brought into use, and the fines were discharged in such a way that much adjacent ground was covered by them. Valuable dredging ground, even now, is being wasted in this way, as also by faulty manipulation of the dredge. The tailings have not been stacked with enough care to keep them well away from the sides of the cut when there is virgin ground on one or both sides.

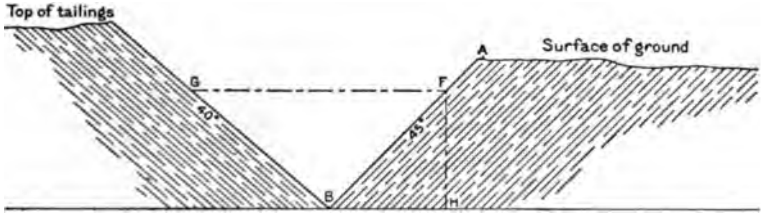


FIG. 35.—Tailings well stacked.

Fig. 35 shows the cross section of tailings so piled as to waste no ground. It is assumed that the side slope of the cut is 45°, but this is the case only when ground is very free. "B" in the sketch is the boundary of the cut on bed rock. "A" is the boundary of slope to 45°. F G is equal to 2 F H. Tailings, as a general rule, should be so stacked that the distance from the tailings to the side of the cut at the surface of the water is about twice as great as the depth to which the dredge is digging.

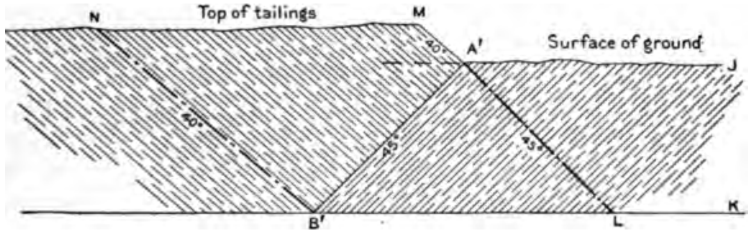


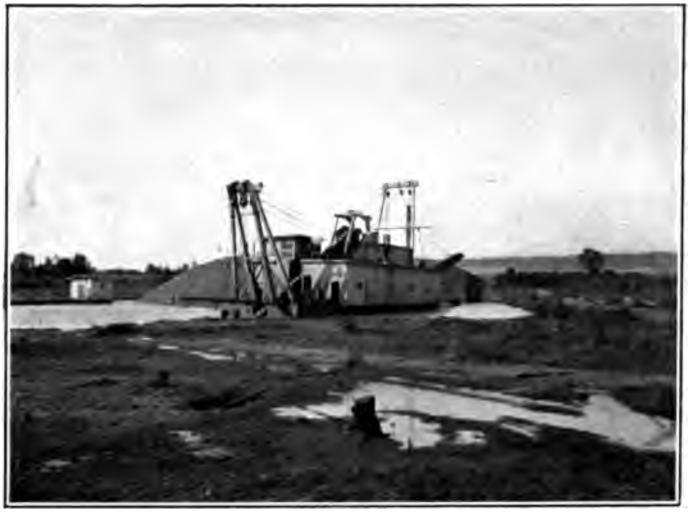
FIG. 36.—Tailings poorly stacked.

A cross section showing the result of stacking tailings badly is indicated in fig. 36. This cut has been filled with tailings to point A', the boundary of the cut at the surface. In excavating the cut, of which A′, L, K, J is a part of cross section, it would not be possible to dredge beyond the point L in the direction of B′ without having to dig tailings in section B′, A′, M, N. If the material in the triangular section is assumed to be loose and to stand at an angle of 45°, angle A′, L, B′, the undredged ground in section B′, L, A′ would be lost.

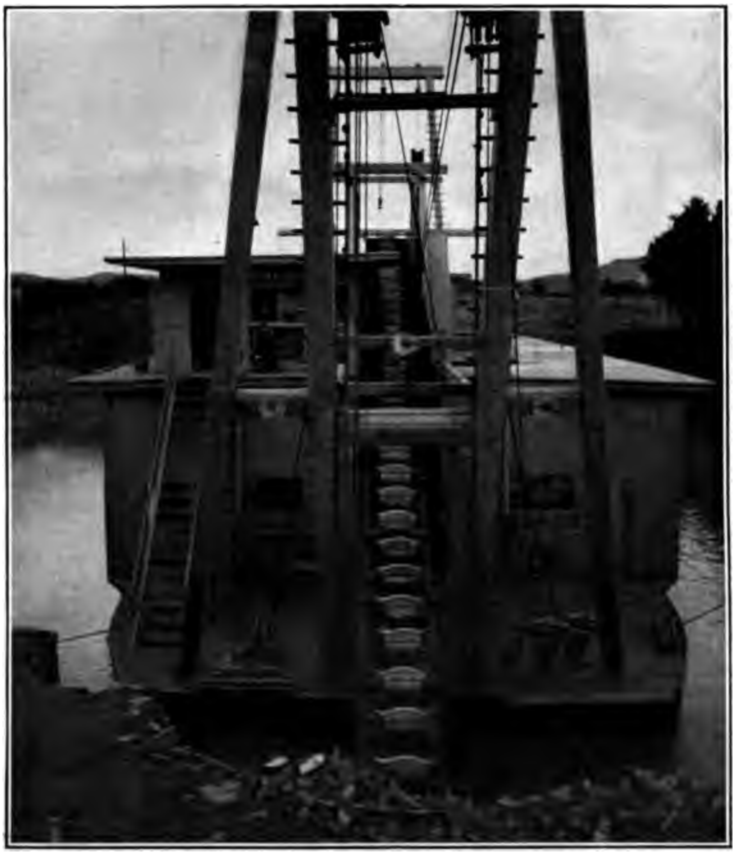
At first the relative merits in the use of a pivotal spud or a headline were considered almost wholly from the standpoint of actual excavation, and the difficulties of disposing of the tailings did not receive nearly the attention their importance demanded. In the opinion of the writer the pivotal spud should be used only by dredges of the double and single lift type whose sluices are sustained by auxiliary scows. Several dredges of these types in Montana, having broken a number of pivotal spuds while dredging a tenacious material, have substituted a system of rope anchorages which does not recommend itself to the writer, as it appears better to make use of stronger spuds. The rigging consists of four wire ropes leading from a high frame at the stern to anchors ashore. The pivotal spud appears to be more effective in holding the dredge when excavating indurated material and prevents its surging and ramming the bank. Attempt has been made to prevent this movement in headline dredges by placing the buckets closer together and thus bringing about greater continuity of contact between the buckets and the dredging surface. This, however, has not been entirely successful. It is possible that the introduction of a number of rollers on the lower side of the bucket ladder and behind the lower tumbler would result in holding several buckets firmly against the material being excavated and prevent surging, as it is probable that this movement, in digging hard material, is caused by the alternate taking hold and letting go of the buckets, as only one bucket at a time is held firmly by the lower tumbler. The buckets behind the lower tumbler dig by their weight only, which is insufficient to force them into hard material. The use of some such device as above suggested, if successful, would probably make it possible for a headline dredge to excavate indurated material with the very important advantages of mobility and the better distribution of tailings.

The use of lower tumblers of more numerous faces causes more buckets to dig at a time and results in less surging. It is possible that the use of a round tumbler of large diameter might entirely eliminate the surging, for there would then be no raising and lowering of the buckets as there is during the revolution of the tumbler of few faces.

The use of the pivotal spud introduces a number of problems in the disposition of tailings. The spud should be so located as to make the radius of the arc described by the buckets in digging as long as possible, thus permitting the excavation of a wide cut—an obvious advantage. This demands the placing of the spud at or near the stern. The



*A.* PRACTICE IN POND DREDGING, OROVILLE, CAL.



*B.* PILOT HOUSE, UPPER DECK, STARBOARD SIDE.

older boats were provided with short stackers and tail sluices, which resulted in the narrow distribution of all the tailings, but more particularly the fines. Under these conditions the stern of the dredge was frequently aground, unless kept clear by the use of a sand pump. This difficulty has been remedied to a certain degree by the lengthening of stackers and tail sluices, but will always be much more serious for the pivotal-spud dredge than for the headline dredge. The headline dredge can use the maximum amount of tailing area. For example, in unusual cases, as in dredging a pay streak of defined boundaries, tailings can be deposited on barren ground outside of the cut. To accomplish the same result with a pivotal-spud dredge numerous changes of position would have to be made, an expedient likely to result in much loss of time and some loss of ground because of the difficulty of relocating the pivotal points. The wasting of unworked ground by the faulty discharge of tailings is well illustrated by fig. 37. In this the distance from the pivotal point to the lower end of bucket ladder excavating at the maximum depth, called the digging length, is 90 feet. Here P $x$ and P $y$ represent the distances from the pivotal point to the point of discharge of the stacking ladder and are called "tailing lengths." Here the discharge is at points marked $z$ and $z'$, which are, respectively, only 18 feet and 5 feet from boundaries of cut, whereas they should be not less than 40 feet away, if the dredge be digging in ground 20 feet deep. It also shows that in some methods of manipulation too long a stacker is detrimental. This illustration is taken from two dredges now in operation.

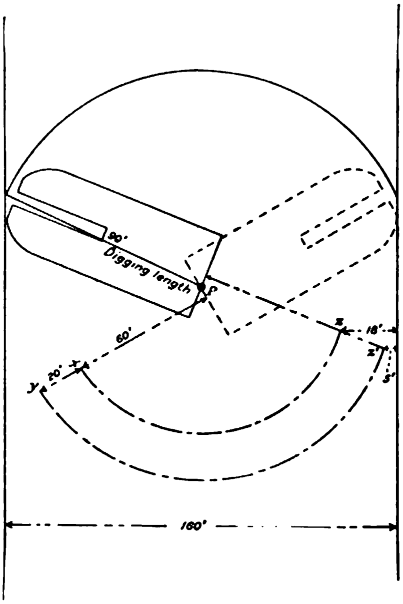


FIG. 37.—Spud digging with tailings piled too near unworked ground.

In fig. 38 a method in general use is shown to prevent the covering with tailings of unexcavated material adjacent to cut. To bring this about it is necessary to move the dredge from P to P′ as often as the distances from C and C′ to $x$, while moving ahead in excavating arc A, B, C, and in the advancing of tailings in arc $x$, $y$, $z$, become so short as to make it difficult, as $x'$ approaches P, and $x$ approaches P′, to move to and dig in the other half of the cut. The changing of the dredge from one half to the other half of the cut is generally made every few days and consumes considerable time. There is, moreover, the constant danger of the dredge not being located with the pivotal point in the right place and thus, when changing from one half cut to the other, there is a consequent loss of ground. In this figure it is assumed that the dredge is digging 20 feet below the surface of a pond on a bank 30 feet deep.

In fig. 39 the result of keeping the lines of advance of the pivotal points too close together is shown. The dumping of tailings at *x′* and *x* would result in grounding the dredge. In this case the distances from *z* and *z′* to sides of cut are unnecessarily great. It is evident that if the lines of advance of the pivotal points are kept too far apart ground adjacent to the cut will be wasted. In general, it may be said that where the material to be dredged is free, the headline method is superior, and also, if by the introduction of some device, surging can be prevented, the headline method would be superior in dredging indurated material, for there the problem of the disposition of the tailings is much simpler. This permits of the use of shorter stacker and tail sluices and, therefore, smaller hulls and lighter construction.

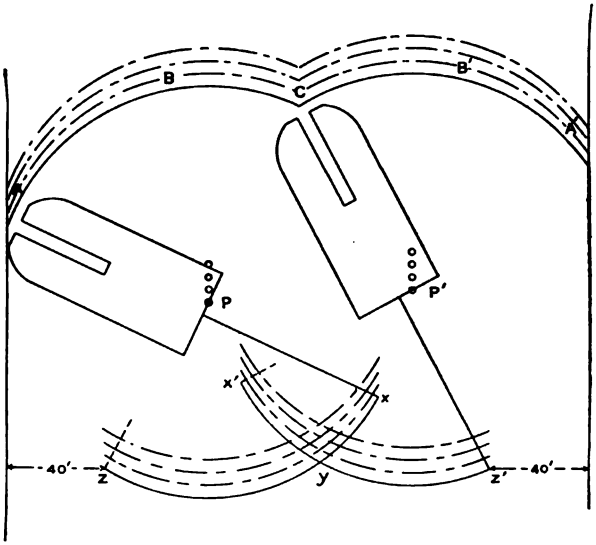


FIG. 38.—Correct spud digging from two pivotal spuds.

There is a marked tendency now toward stronger hull construction, and longitudinal bulkheads or fore and aft keelsons are often introduced. Traverse water-tight bulkheads at the bows would be advantageous in dredges working in torrential rivers where there is danger of collision with floating trees; or, in rivers like those in Alaska, where ice may be encountered.

In some instances machinery too heavy for the size of the hull has been installed, and, as a result, there has been a lack of stability of the dredge, with consequent disturbance of the grades of the gold-saving devices. Hulls are now generally made with overhanging houses, which give more deck room and permit of the location on board of a small though very convenient shop. The crowning of decks has resulted in cleaner and drier dredges.

Stronger gauntrees have been introduced in the late installations. The bow gauntree, at first considered merely as a support for the outboard end of the digging ladder, is made much heavier, as it is now designed also to tie together the bow pontoons.

The main gauntree, on which are sustained the upper tumbler and its driving gear and the inboard end of the digging ladder, is made much stronger than formerly, particularly when the upper tumbler has a gear drive and a good alignment becomes therefore essential. In dredges recently constructed the main gauntree has been made the vertical post in longitudinal and transverse trusses which tend to prevent the distortion of the hull, due to the great weights of the digging and tailing machinery, which are sustained respectively at extreme bow and extreme stern. Stern gauntrees have not been much modified, but have been strengthened and made somewhat higher, so as to sustain longer stackers.

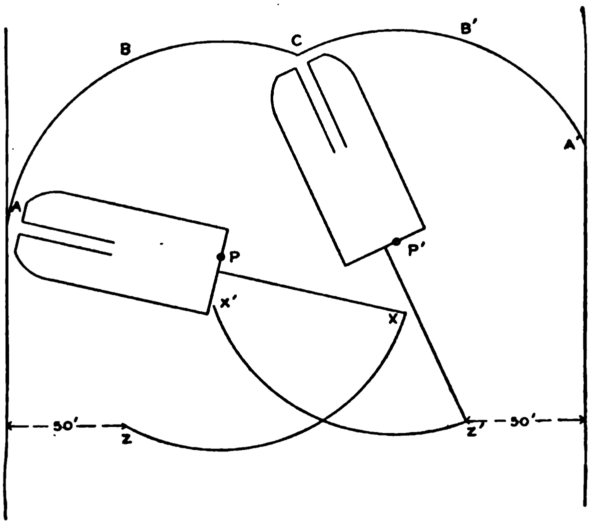


Fig. 39.—Sketch showing danger of grounding boat in tailings when pivotal points are too close together.

The introduction of some form of derrick or crane, at the bow, to move stumps and other obstructions and to hoist machinery, is now common. The use of traveling cranes over driving and other machinery is not so general as it should be, for such devices save time and labor when, as is frequently the case, renewals or repairs are necessary. The machinery is now generally housed.

The location of operating levers, controllers, and switches is still a mooted question. Some operators prefer them on the lower deck close to the driving machinery; others in the pilot house, which is located near the forward end of the upper deck. In some installations they are placed in a pilot house just aft of the upper tumbler. With the first arrangement the winch man is among the driving machinery and can watch it, but unless he leaves his station he sees little of anything else and gets a poor view of the whole length of buckets. With the second arrangement the winch man can see the buckets better, all bowlines, but little else. With the third arrangement the winch man sees the buckets, the lines at bow and stern, the hopper, the screen, and enough of the gold-saving apparatus to know whether the riffles are clear, and with the aid of mirrors can also see the stacker and tail sluices. The objection to this arrangement and, in a measure, also to the second is that the winch man is isolated and therefore inaccessible (see Pl. XXXV, *B*, p. 184). The type of dredge and height of upper tumbler should determine the position of the operating levers, controllers, and the switches, but in general it would appear that the advantage would lie with a location behind the upper tumbler.

Loss of time has always been a serious factor in the use of gold dredges. The first installed in the United States operated less than 50 per cent of the time. This was due to several causes, but principally to the time consumed in making repairs and renewals and the difficulties brought about by the faulty arrangements for disposition of tailings. Minor causes, such as lack of cranes and derricks, can be overlooked, being unimportant by comparison. The writer, however, has seen the removal of troublesome stumps consume hours when, almost without exaggeration, it may be said that minutes should have sufficed for this purpose. It is noteworthy that many operations are hampered by lack of a sufficient number of lights for doing night work on or near the dredges.

All dredges should be equipped with devices for hoisting and moving the driving, screening, and pumping machinery, but these are now often lacking. The belting, both for stacking and driving, has caused much loss of time—a result due both to the poor quality of belts and to the attempt to make belts do work for which they were not intended. Friction clutches, hoppers, and screens of bad design and inaccessibility of parts of equipment have also brought about a serious loss of time. This is specially true of renewals of such parts as baffle plates of hoppers and perforated sheets of screens.

Cleaning up also consumes much time, and any arrangement that will permit continuous running during clean-ups of riffles charged with quicksilver will be of great value. Of late some managers have employed rather large crews to minimize lost time, though many dredges are still operated with but two men per shift.

The now general use of variable-speed motors on electric dredges has had good results. Constant-speed motors on headline dredges working in indurated material had brought the headline method of dredging into bad favor, whereas variable-speed motors permit a chain speed suitable for indurated material, with less surging and breakages. Under these conditions about the same amount of material is excavated, as a slow chain speed with low theoretical capacity has often in tenacious gravel a greater actual capacity than a high chain speed with high theoretical capacity. This is due to the incomplete filling of the buckets when running at too great a chain speed.

In Alaska, where much frozen ground is encountered, the conditions are not favorable for the successful operation of gold dredges, especially as the long winter reduces the number of working days to about one hundred and thirty a year. Nearly all the deposits of auriferous alluvions are frozen, though some areas, notably in the beds of rivers, are unfrozen. Such unfrozen zones are easily dredged and large volumes can be cheaply handled. The depths to bed rock are moderate, but the bed rock is generally unfavorable for easy excavation, and the ground is characterized by a great concentration of values near and in it. Lateral concentration also is great and the pay streaks are often narrow. High costs of installation and operation are other drawbacks in regions far from the coast, to which freight charges are high. Considerable areas of so-called "worked out" ground in this northern field, particularly in the Klondike district of the Yukon Territory, would furnish excellent dredging ground if the material were not frozen. Steam thawing now costs about 40 cents a cubic yard, but a cheaper method of thawing would make much ground available for highly profitable dredging. It is probable that if wide cuts were made the banks might thaw as rapidly under exposure to the sun as they could be removed by the dredge. Another factor which may be of importance is that ground which has once been worked does not freeze to any great depth. It is evident that the chief problem of gold dredging in this northern province is in the thawing of the material, for many of the other conditions are favorable for profitable exploitation by this means.

Had gold dredges been used in the North when the first mining was done on the richer creeks much more gold would have been recovered and the cost would have been less, for the earlier methods were exceedingly wasteful and costly. The overburden of muck could have been easily sluiced off and the thawing of the underlying gravels would have been easy when aided by exposure to sun and air.

In this northern region, because of the small vertical section and absence of much fine sand in the gravels, dredges with short digging ladders and stackers can be used. This equipment demands only small, light hulls and machinery. Large volumes of thawed material can be handled, but screening devices for saving coarse gold are essential. Bucket rather than belt stackers are preferable, for ice on the faces of the driving pulleys would cause slipping. The climatic conditions demand that the dredges should be completely housed, and that those driven by electricity should be equipped with electric heaters to supply hot water to the gold-saving appliances and thus permit continuing operations after cold weather has put a stop to hydraulicking and open-cut mining.

Open connected buckets will do better work in the heavy bed rock usually encountered in Alaska, and teeth on bucket lips will probably be effective. The use of several powerful digging teeth in the bucket chain, instead of toothed buckets, to excavate heavy bed rock, was suggested by a manager of wide experience, and has worked well in several instances. A larger factor of safety should be allowed in designing dredging machinery intended for Alaska, and strenuous efforts made to minimize lost time in every feature of the construction, so that full advantage may be taken of the short available season.

## SLUICES AND GOLD-SAVING APPLIANCES, EXCLUDING HYDRAULIC OPERATIONS.

Creek miners in the Klondike and Alaska placer fields have met, with extraordinary vigor and a considerable amount of success, the peculiarly difficult conditions attendant on mining operations in the Northwest. Inventive genius has been called largely into play, since, except in parts of Seward Peninsula, hydraulic mining in working the creek deposits is not practiced. It is evident, however, to one who visits the Klondike district, that the methods there in vogue for working the rich creek deposits have been developed with special attention to the economical mining and conveying of the material to the sluice, while the washing of the gravel in the sluice is not, as a rule, conducted with a view to the saving of the greatest economic amount of the gold. Whereas in the hydraulic-sluicing methods the benefit of long experience has resulted in generally commendable practice, the smaller hand and mechanical creek operations frequently exhibit gross carelessness in the matter of gold-saving appliances.

The method of shoveling by hand into a string of sluice boxes is naturally the one first tried by the miner in a remote district, working in shallow ground, after he has passed the panning and rocking stage. In these operations the cost, even under present northern conditions, varies from $1.25 to $2.50 per cubic yard (averaging $1.63), the capacity per man per shift averaging 5½ cubic yards. The most primitive appliances are the most economical. From three to six boxes, 12 or 14 inches in width by 12 inches deep by 12 feet long, on a grade of 6 or 7 inches to the box length, fitted with 6-foot 3-inch pole riffles made of saplings, form the customary rig in the interior where timber is at hand. From 30 to 60 miner's inches[a] of water are used. Operations of this kind, where from three to twenty men shovel in, are to be found on all the gold-bearing creeks exploited in Alaska, though in the Klondike they have been largely supplemented by methods employing less hand labor. As none but rich gravel can be so worked, exigency permits a loss of fine gold. It is rare that placer miners will admit that they are losing gold, but it is safe to estimate that in the interior, where two to five boxes are in use for saving, and where drop-offs are not used, or are only such as are caused by the telescope connection of the boxes, from 10 to 20 per cent of the gold lifted into the boxes is allowed to return to the creek bed.

It would seem that heavy losses must occur in connection with the sluicing of the winter dumps taken out in drifting operations. The strings of sluice boxes are erected at as small an elevation as possible, in order that the greater proportion of the material will not have to be rehandled when the spring sluicing is done. Pl. XXXVI, *A*, shows one method of dumping, in the winter, over previously erected sluices. Boards are laid over the sluices, and when sluicing is resumed the water is turned through the sluice, and, beginning at the lower end, the boards are successively removed. As much of the gravel as possible is caved in, sometimes with the assistance of a nozzle, and the remainder is shoveled, wheeled to, and dumped into the sluice. Small bunkers or hoppers are sometimes built over the sluices, but no hoppers of large capacity, like those in use in Plumas County, Cal., were seen in the north. On Anvil Creek, in Seward Peninsula, a large winter dump was handled in this way, with the exception that those portions which could not be caved to the sluice were conveyed to it and dumped in by means of horse scrapers. Though loss of gold may be permissible in primitive operations of small capacity, it should become proportionally less when larger mechanical installations are made and the capacity of the plant is increased.

One of the early difficulties which the miner in the interior encountered was the presence of sticky clay and mud in the rich pay dirt. The difficulty was partly overcome by the introduction of the mud box, or puddling box, which was set in the middle or at the upper end of the string. Into this the men shoveled, or a bucket or car dumped. Pl. XXXVII, *A* (p. 194), shows the form of the mud box used in the Klondike, and fig. 40 shows its position in the line of boxes. Its grade is generally made steeper than that of the rest of the string; 12 inches is common. The services of an extra man as stirrer, who also forks out the large stones, are required.

Where men shovel into boxes the mud box is used merely as a wide part of the sluice. In larger plants, where buckets, cars, or derrick skips dump into the mud box, a platform, inclined at an angle of about 50°, 9 feet square, and built up of timber floored with rough scantling, is erected on the side of the box for the gravel to fall on.

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[a] The term miner's inch used in this paper signifies an amount of water equivalent to 1½ cubic feet per minute.

The capacity of the sluice is cut down rather than increased by the use of the mud box, and the expense is increased by the cost of the man. A greater saving of gold is made, but at best the operation is expensive and of small and variable efficiency. In some shoveling-in operations the use of the mud box is advisable, but where mechanical self-dumping buckets are used, it is possible that some other form of agitator might be advantageously employed.

The developments of the open-cut and drifting methods of gravel mining have necessitated an enlarging of the sluicing capacity. With the enlarging of the capacity, however, there has not been a proportionate improvement in the construction of the gold-saving appliances. In other words, instead of drawing on the experience of the hydraulic miner and the dredge miner for the adoption of gold-saving methods, the creek miners of the Klondike have continued the method of the long, narrow sluice used for shoveling-in operations, amplifying its error and suffering the inevitable losses of fine gold which its use entails.

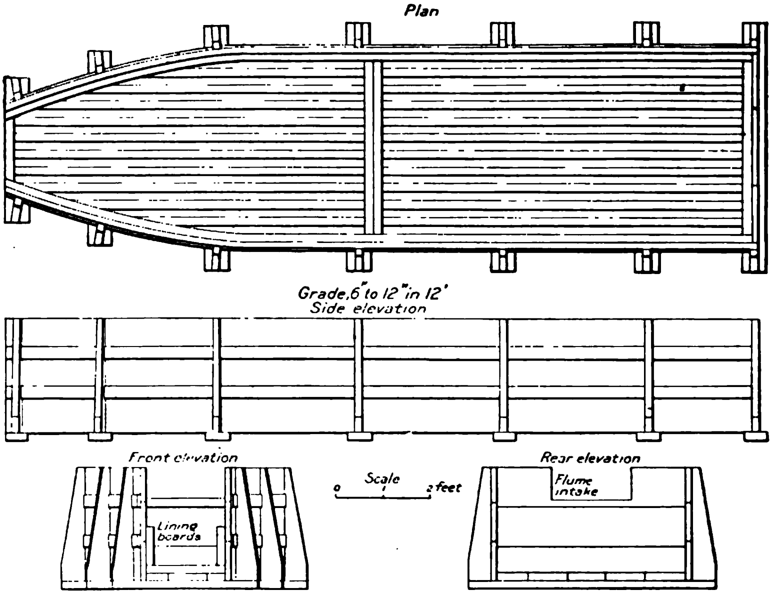
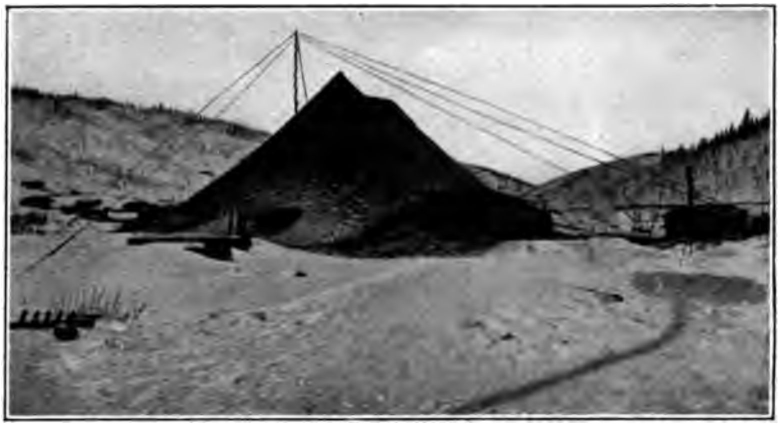
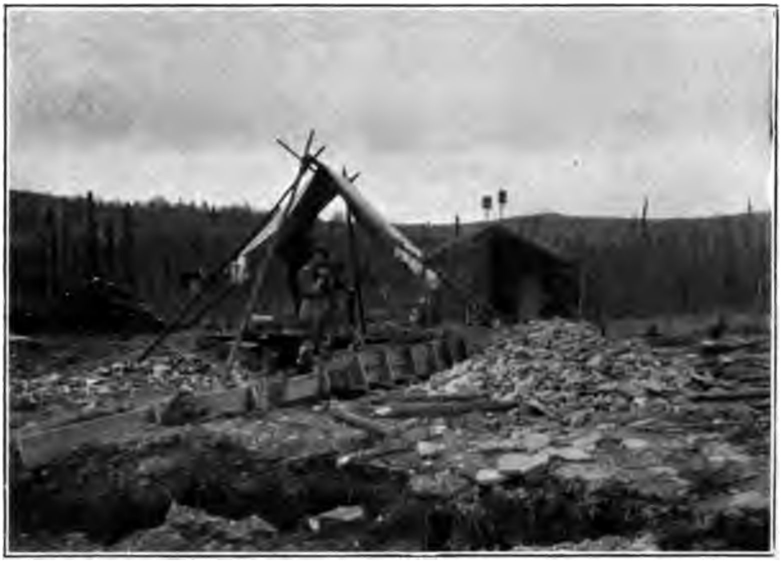


FIG. 40.—Mud box.

The average capacity of a small placer operation where hand labor is employed is 40 cubic yards a day of ten hours. Grant that the narrow sluice of 36 feet in length with pole riffles is most economical for the needs of such a mine. Now take an average summer drifting



*A*. WINTER DUMPS, SHOWING PREVIOUSLY ERECTED SLUICE BOXES, KLONDIKE.



*B*. MUD BOX, USED IN STRING OF BOXES.

plant, where the gravel is dumped into the mud box by means of the cable tram and self-dumping carrier, like that illustrated in Pl. XXXVII, *A*. The capacity is 175 cubic yards in twenty-four hours, and the cost is approximately $1.50 per yard. After it is hoisted from the shaft the material is elevated to a height of 25 feet above the surface of the ground. Water is pumped to this height for sluicing. The sluice consists of a mud box 16 feet long and 30 inches wide, on a 12-inch grade, tapering to the 14-inch sluice boxes which follow. There are eight of these, set on grade of 10 inches to 12 feet, furnished with pole riffles, which last three weeks only and cost $3 per box length to renew. The man forking in the mud box costs $6.50 a day of ten hours. At this plant the gold is in part very finely divided, and it is impossible to believe that the sluice in use is operating with economy.

Proof of the losses now going on in the Klondike was seen on a neighboring creek. At a plant somewhat larger than the one above described, where 240 cubic yards a day were handled at a cost of $1 a cubic yard, a small undercurrent had been installed, at the end of ten 16-inch boxes, 12-inch grade, pole riffles. The undercurrent was fed through a small iron grizzly, and consisted merely of one 16-inch sluice box, 12 feet in length, with a riffle of cocoa matting and expanded metal. It cost $20 to construct this device, which was saving an average of 5 per cent of the product each day. A sample of the gold was taken, and although some of it is too fine for handling, such particles as could be weighed and counted gave a result of 280 colors to the cent, the gold being worth $15.60 per ounce. Gold of finely divided but never flaky character was seen in all the large producing creeks of the Klondike, and at the new Fairbanks district of Alaska.

The plants above referred to represent the average capacity of the creek mines of the interior of Alaska, where the hoisting of material, and frequently the pumping of sluice water are necessary. It costs from $3,000 to $5,000 to rig up such plants, which are used for three seasons or longer. To install a washing plant in such a case as the above would add little to the first cost, and the additional expense would probably be justified by the results.

Before entering on suggestions as to the use of washing plants, I wish to emphasize the fact that the methods of sluicing in use in Alaska, especially in the Birch Creek, Fortymile, and Fairbanks districts, and to a certain extent in Seward Peninsula, have been, and will continue to be, influenced by the Klondike developments. Though many of the methods developed in the Klondike are excellent and are worthy of imitation in any country where conditions are similar, at the same time the Alaskan miners should note the wrong principle of the primitive sluice box which has been continued there. The entire absence of screening or use of grizzlies, other than the rough, expensive hand method employed, renders valueless much ground which could be worked. A plant costing $5,000, designed to thaw, excavate, hoist, convey, and wash 150 cubic yards a day at a cost of $1 a yard, could, by an addition of from $500 to $2,000 to the first cost of the plant and an addition of 10 cents per yard to the cost of washing, recover at least 10 per cent more value in gold from a cubic yard of material.

It may be said that the fine gold found in the Klondike does not occur in the Alaska creeks. This is disproved by experiments made on small parcels taken from pannings on the various creeks. Fine gold from Fairbanks Creek, in the Tanana district, runs 500 colors to the cent, the gold being worth $17.70 per ounce. A small proportion of gold from Ophir Creek, in Seward Peninsula, was found to run 170 colors to the cent, the gold being worth $18.50 per ounce. No means of knowing the proportionate amount of this gold is possible, as with the methods at present employed very little of it is recovered. Five per cent is considered a fair average of the total gold represented by the above fineness of division. This gold, which is not flaky in spite of its fine character, could easily be saved with the proper appliances, and there is no doubt that much coarser gold is lost.

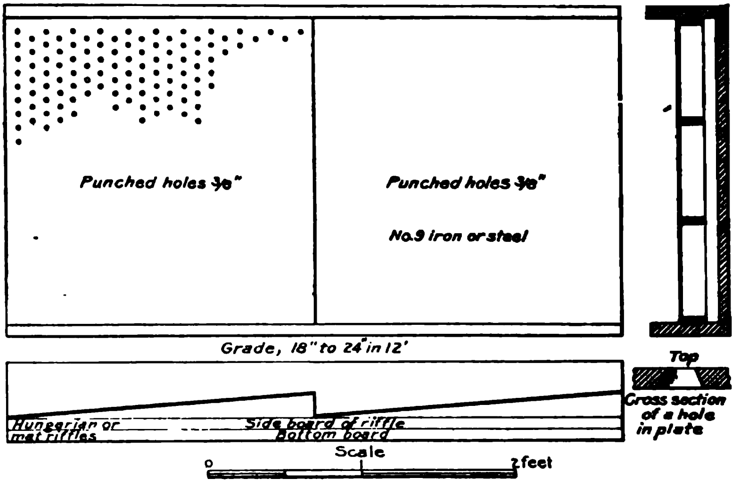
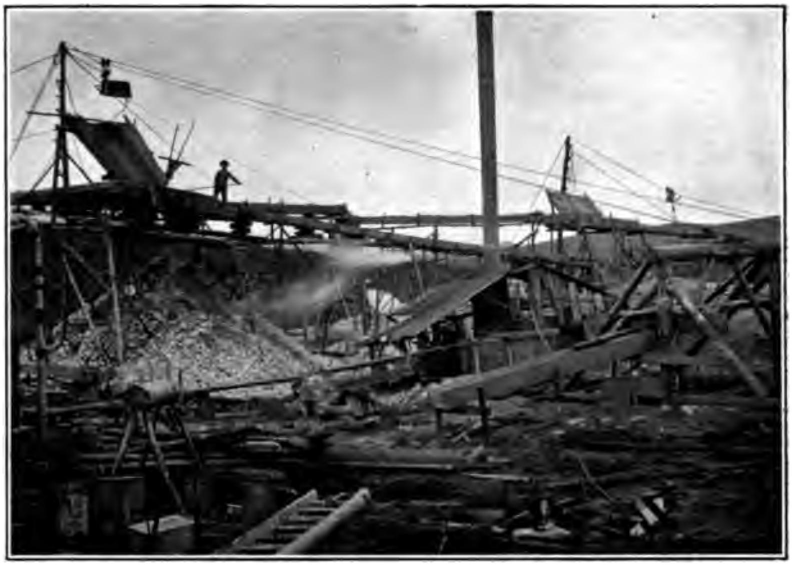


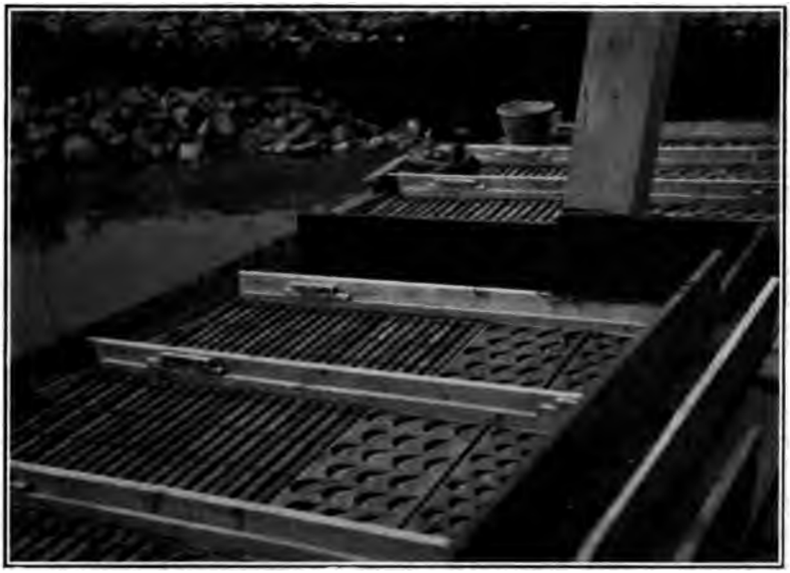
FIG. 41.—Modified Caribou undercurrent.

The question will be asked, Can a cheap and efficient washing plant be installed, where gravel must be elevated by power, which will not add greatly to the cost, and the parts of which can be easily and quickly made with the materials at hand?

On the principle that gold is best saved in the thinnest sheet of water which will carry the tailings away, it is evident that for fine gravel a wider sluice is better than a narrow one. In narrow sluice boxes,



*A.* MUD BOX ON GOLD RUN, KLONDIKE.



*B.* OROVILLE RIFFLES USED AT ATLIN, BRITISH COLUMBIA.

full or nearly full of water running with great velocity, fine gold will be carried along. Lengthening the sluice will not help matters. In fact, a short sluice with drops or undercurrent attachment is frequently more effective than a long sluice without it.

In a given case assume that the ordinary 14-inch boxes are widened to 24 inches, and that 5 boxes, with a grade of 9 inches to the box, fitted with the ordinary sapling-pole riffles, as at present, are succeeded by 3 boxes with the form of screen represented in fig. 41, the grade being made adjustable by means of blocking. An addition of 19½ inches to each of the 3 boxes would compensate for the loss of grade consequent on the uptilting of the screens. The whole drop in the string of boxes would be 103.5 inches as against 108 inches were 12 boxes used with 9-inch grade. Under the punched iron plates riffles of one or more kinds as described below, charged with quicksilver, should be used. The riffle shown in fig. 42 will be found satisfactory. Mats, plush, or blankets may under certain conditions be found more economical. A short transverse table following the last box, arranged under a grizzly from which the large material is discharged to the dump, will enable the operator to determine whether losses are occurring in the main sluice and screening boxes.

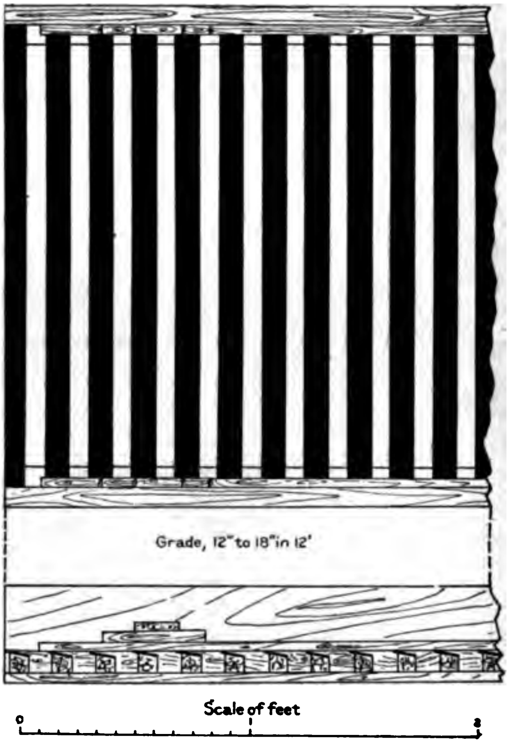


FIG. 42.—Improved Hungarian riffle used at Oroville.

The above installation is in principle a series of undercurrents combined in the main sluice. Its advantages of cheapness, adaptability to conditions, and simplicity recommend it. No extra power or water is required. On the other hand, the use of the mud box is not obviated.

Regarding the use of punched plates in working New Zealand gold-bearing marine deposits, Mr. H. W. Young[a] says:

> The best size of perforation for hopper plates has been a matter for experiment by myself and others, variations from one-fourth inch to five-eighths inch having been put

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[a] Report of New Zealand Minister of Mines, 1902, p. 21.

to working tests. It is proved that holes of less than three-eighths inch diameter unduly limit the discharge through the plates, are liable to choke, and that there is no advantage whatever to be gained by their use. The discharge through seven-sixteenths inch holes is sufficient in amount, and the fine shingle particles which pass through with the sands and seldom exceed one-fourth inch in diameter are not troublesome in their size and quantity. In fact, many consider that they are of benefit in keeping lively the sands on the tables. With holes exceeding seven-sixteenths inch in diameter, the size and quantity of small shingle become excessive, and the water passes away so rapidly as to prevent material from being carried forward over the plates.

The above remarks are of somewhat general application for the use of punched iron screens in any form of undercurrent where it is assumed that a portion of the gold, including any nuggets which may occur, has been previously saved. In northern deposits where dredges have been installed, as in the Klondike and on Stewart River, punched iron screens used in revolving trommels have large holes, up to 1½ inches, even when a tailings stacker is used, while the average size in Oroville dredging practice is three-eighths inch.

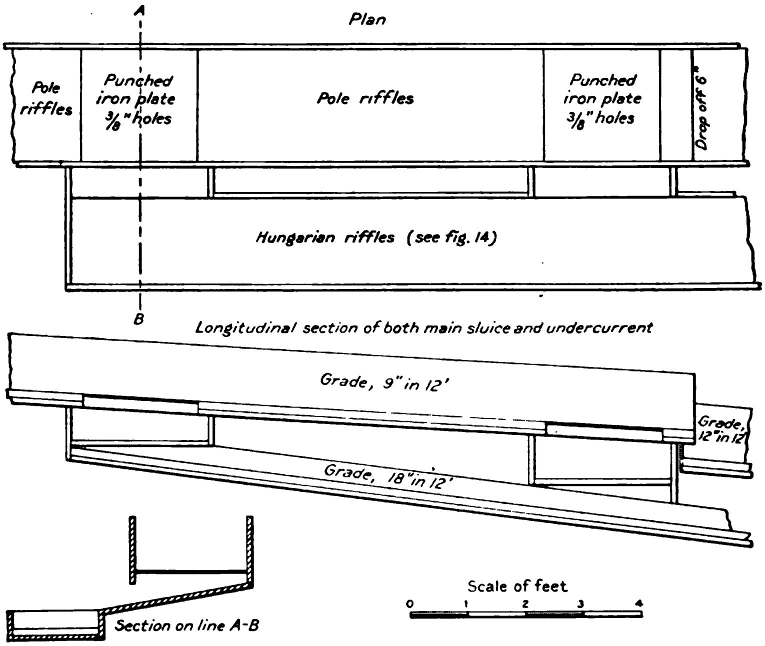
A
Plan
Pole riffles
Punched iron plate ⅜" holes
Pole riffles
Punched iron plate ⅜" holes
Drop off 6"
Hungarian riffles (see fig. 14)
B
Longitudinal section of both main sluice and undercurrent
Grade, 9" in 12'
Grade, 12" in 12'
Grade, 18" in 12'
Section on line A-B
Scale of feet
0
1
2
3
4

FIG. 43.—Kulibinka sluice and undercurrent used in Siberia.

A screen installed in one box of the main sluice of a plant on Ophir Creek, Seward Peninsula, consisted of a number of longitudinally disposed round iron rods, acting as a grizzly, fitted above the bottom of the box, the gold being saved on mats below. This box, which was placed at the end of a 120-foot sluice, was said to save much fine gold.

When it is desirable to introduce the principle of the undercurrent, separate from the main sluice, the following device, adapted from one used in Siberia, as described by Mr. E. D. Levat,[a] may be adopted. Its capacity is given as 172 cubic yards per shift. The principle of the undercurrent is here introduced, but the discharge, of both coarse and fine, is to the same heap. This contrivance uses no power, and the method of handling accumulations of tailings will be more expensive than in plants already in use. Where the steam scraper is so generally employed, however, as in the Klondike for handling tailings, the innovation of the modified "kulibinka" here figured (fig. 43) would not add over $1,000 to the expense of installation, while the efficiency in saving the values would be greatly increased.

It has been shown that the Klondike sluice generally necessitates a man forking. The object of this forking is to take out the stones, from 6 to 18 inches in diameter, after their surface is washed. Mechanical devices for accomplishing this would be advantageous. Experience with gold dredges has proved that the revolving screen or trommel, inside of which play powerful jets of water, accomplishes this screening process most successfully. The trommel is, however, expensive, and its various parts and castings must be specially made at elaborately equipped works. Therefore it is worth while to consider whether simpler and cheaper devices will not accomplish nearly as good results for the Alaskan miner.

A plant to accomplish good washing results with sticky clay and gravel, and which can be built of materials at hand, is shown in fig. 44. This type of plant, founded on the idea of the Siberian pan, has a capacity of from 100 to 200 cubic yards in ten hours, and can be built in the winter months. Assuming that steam power is already at hand, it requires no outside material beyond the iron shoes and simple castings and the punched steel plate which forms the floor of the pan. Its operation will require 10 horsepower, and if the material is conveyed to it by the self-dumping carrier, the services of two men are sufficient to take care of the tailings. The machine will not only break up and thoroughly wash clayey gravel, but with properly arranged tables below will save the bulk of the fine gold which has been set free from its matrix. The cost of a pan of the dimensions here figured will not exceed $2,000, including the tables and sluices. The device for automatically clearing the bottom of the pan of large stones is not used in Siberia, where hand labor is cheap enough to dispense with it, the large stones being periodically removed by the lifting of gates in the periphery of the pan. The amount of water used in such a machine does not exceed ordinarily 125 miner's inches.

The drawing of this machine is made diagrammatically, since the manner of its construction will depend on local conditions. A four-armed casting, keyed to the shaft and bolted to the horizontal timbers, is advisable. As used in remote districts in Siberia, the central shaft is frequently a wooden beam, and instead of the iron shoes, heavy stones dragged with chains, as in the arrastre, may supplant them.

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[a] Levat, E. D., L'Or en Sibérie orientale, Paris, 1897.

The above suggestions are made with reference to working the rich gravels of the shallow northern placers. Gravel containing less than $2 to the cubic yard is rarely worked by the method of the cable and

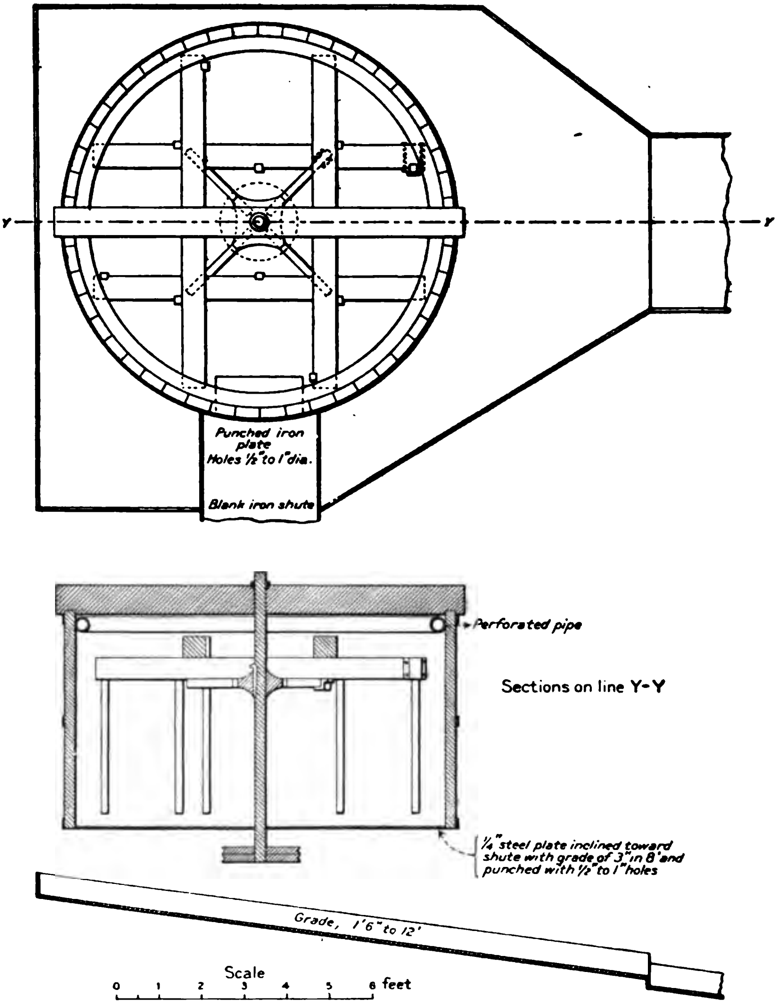

FIG. 44.—Modified Siberian pan for handling clayey gravel.

traveling bucket. Lack of natural grade makes it necessary that the miner shall elevate his material, and the impossibility of getting water under head makes the use of steam power imperative. The point is that when such elevation is attained, it should be made available to thoroughly wash the gravel. At present this is not done. Perhaps the suggestions here offered may be of benefit to some operators.

In the construction of washing plants of a larger and more expensive kind, the operator has the benefit of the experience developed in gold dredging. This now rather important industry has brought into service devices for gold saving, in the use of which one of the main objects is to utilize to the best advantage all available vertical and areal space.

The employment of large mechanical excavators for placer mining has a field in the Northwest. Such operations necessitate one or more permanent washing plants to receive the gravel from each machine. It has been demonstrated that the dredge is the only form of excavator which can economically transport its sluices as it moves. Therefore a plant, situated as safely as possible with reference to danger from floods, and economically with reference to tramway, dump, and water supply, must be constructed frequently at a considerable expense.

The shaking screen, although it has received a thorough trial on gold dredges, does not find as much favor as the trommel. It is not impossible that modifications of the principle of the shaking screen and of the shaking table may be developed which will act more efficiently in saving gold than the trommel. Mr. Félix François[a] has recently figured and described a "shaking sluice-box" system of gold saving, for installation either in stationary washing plants or in dredges. He claims a very high percentage of saving, and as additional advantages the elimination of the use of quicksilver and the employment of a small amount of water. He does not give the cost of the plant, however, nor any actual results of its operation in practice.

The use of the trommel in a stationary washing plant is illustrated in Pl. XII, *A* (p. 80), a plant erected in the Klondike in connection with a steam shovel and incline operation. A short description of this plant is appended, but it should be understood that for the average miner the installation of such a plant is impracticable on account of first expense and the difficulty of getting the complicated machinery. The plant used 125 miner's inches of water, led by a ditch from Bear Creek; the capacity was said to be 500 cubic yards in ten hours. The material elevated to the platform at the upper end of the trommel was dumped into a hopper feeding the trommel. The water was led into the lower end of the trommel and fed through a perforated pipe. The largest holes in the revolving screen were 1 inch in diameter, and all oversizes passed through and into the hopper below the lower end, whence the tailings were hoisted in a self-dumping carrier, on a cable, for a horizontal distance of 200 feet and a vertical distance of 60 feet. The fines passed over 80 square feet of riffle tables, floored with expanded metal and cocoa matting, on a grade of 12 inches to 12 feet, followed by sluices with iron Hungarian riffles. Pl. XII, *B*, shows the expanded-metal form of riffle table, which has given good satisfaction. The fines were, after passing out of the 96 feet of sluice boxes following the tables, elevated by a steam scraper to a pile 200 feet distant and 15 feet high. The expense of installation of such a plant will be not less than $5,000, and will more likely be $10,000 in any part of the interior of Alaska.

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[a] Bull. Soc. de l'industrie minérale, vol. 3, 1894, p. 785.

In arranging a number of gold-saving tables to receive the discharge from a screen, great care should be taken to distribute the material equally to these tables, so that the duty of each may be the same. This in the best Oroville practice is done by leading a small sluice trough from the main receiving sluice beneath the screen to each of the tables. In case of a second sizing, as in the Atlin dredge described below, the distribution is necessarily accomplished by a series of grizzlies in the main sluice. If small ducts or troughs are used, they should be provided with gates, the whole made of wood, like those used for distributing the pulp to the tables in concentrating mills. The attempt to distribute the fines directly from the trommel by means of iron gates is considered less satisfactory.

Riffles for the saving of fine gold in sluices are of many kinds and are of very ancient origin. Humboldt (Asie Centrale) refers to the method in use of working the placers of Colchis—that of employing wool in the sluices—as a possible explanation of the legend of the "Golden Fleece." It was in fact known that the kings of Imeret in the eighteenth century used wool for collecting gold in Tskinitskali and Abacha rivers in the Caucasus, while Turkish gypsies use goatskin for gold saving on Belichta River.

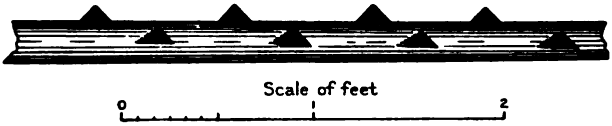

FIG. 45.—Pole riffle fitted with knives for breaking clay.

The pole riffle made of saplings, with or without strap iron nailed to the top, has long been in favor in small placer operations in the United States, and is to-day employed in the primitive shoveling-in operations throughout the northern territory. An improvement on this riffle, simply made, aiding in the disintegration of clay, was seen by Mr. Frank L. Hess in the Rampart district of Alaska, and a sketch furnished by him is shown in fig. 45. Small squares of sheet iron one-sixteenth inch by 2 by 2 inches are driven cornerwise into the poles.

A development from the wooden pole riffle is the iron or steel rail, laid longitudinally in the sluice box. One type of rail riffle used in Seward Peninsula is shown in fig. 32. It will be noted that the T lies with its horizontal extension uppermost, that the rails are joined in gangs of four, and that a locking device is employed to prevent the gangs from coming up. Rail riffles of this kind are primarily intended for hydraulic operations and give excellent satisfaction, but can be recommended for creek placers only where the capacity warrants expensive installation.

At the present day the saving of fine gold is receiving marked attention on gold dredges, as the product of these machines frequently consists largely of gold in an exceedingly fine state of division. Fig. 42 and Pl. XXXVII, *B*, represent a sluice table now in use at Oroville, Cal. The grade of the table on the dredge at Atlin, British Columbia, is 12 per cent, corresponding nearly to 18 inches to 12 feet. These tables are set transversely beneath the main sluice, which is fed directly from the trommel, the main sluice having a grade of 12 inches to 12 feet. The quicksilver charge of the sluice and table is 200 pounds. The largest material passing over the sluice is 4 inches in diameter, while grizzlies do not permit over ½-inch sizes to go over the undercurrent tables. The total area of sluices and tables is from 600 to 1,000 square feet, from 500 to 700 miner's inches of water being used for an amount of material which will probably average 1,500 cubic yards per twenty-four hours.

The type of riffle here figured is used extensively at Oroville at present. It is stated by one of the operators who has experimented with the gold in that field, that from 15 to 20 per cent of the gold recovered with quicksilver, using this riffle, will pass 150-mesh sieve.

The expanded metal and cocoa matting riffle is also used with success at Oroville.

The Oroville gold contains a much larger proportion of fine colors than the northern fields. From the undercurrent sample of Klondike gold above described, assuming that this represents 5 per cent of the total recovery, screening tests appear to indicate that under 1 per cent of the Klondike gold, and under 2 per cent of the Fairbanks Creek gold, will pass 150 mesh.

The fineness of the gold on Sulphur Creek is shown by the table below:

*Fineness of gold from Sulphur Creek.*

| Mesh. | Per cent. |
|---|---|
| Under 150 | 10 |
| 150–100 | 60 |
| 100–80 | 20 |
| 80–60 | 10 |

Moreover, the gold, whatever its fineness of division, is generally round and shot-like and not flaky. Under such conditions, in view of the Oroville experience, losses such as undoubtedly occur in the northern practice are inexcusable.

The riffle shown in fig. 46 is designed to hold a divided sheet of quicksilver. Another form is made by boring 1½-inch augur holes to the depth of one-half inch. Some operators claim that a recurring quick jar imparted to the fine gold tables by means of an eccentric or other device aids the saving of the gold. The efficiency of this principle is questionable.

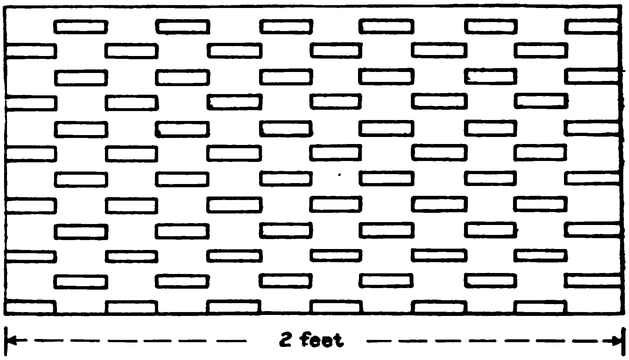

FIG. 46.—Riffle to hold divided sheet of quicksilver.

On the Snowflake claim, a bench claim, worked by drifting, situated between Anvil and Dexter creeks, much of the gold sluiced from the winter dump was porous, occurring in lumps one thirty-second to one-fourth inch in diameter, resembling dentist's gold. A peculiar riffle shown in fig. 47 was devised, consisting of sawn blocks nailed to riffle strips.

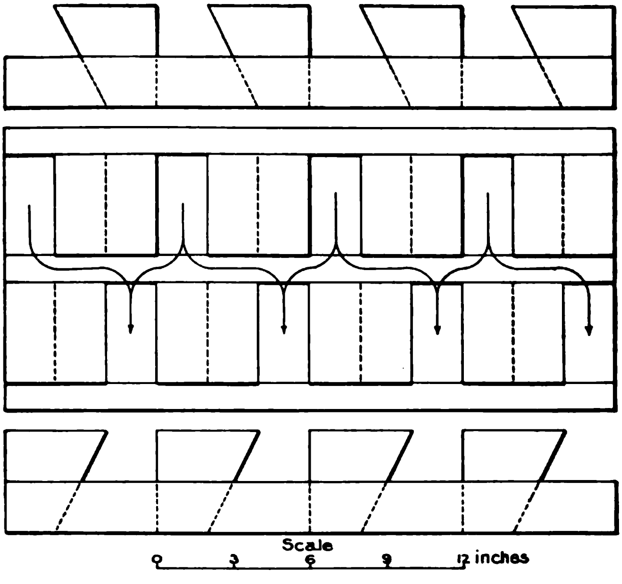

FIG. 47.—Special block riffle for saving porous gold.

This riffle used in 14-inch boxes, on 8-inch grade, with 50 miner's inches of water, was said to be the only one of many tried which would catch this gold.

Fig. 48 shows an excellent iron riffle used in the smaller sluices of Seward Peninsula for saving gold of average fineness. The castings are light, can be easily handled, and can be set in the sluices so that the long dimension of the slots lies either transversely or longitudinally. The longitudinal arrangement has been found to be the better.

The use of quicksilvered copper plates is not likely to give increased saving in Alaska placer operations. Blankets, mats, or other fabrics are not generally used on account of their expense.

Mr. H. W. Young[a] has designed for the Waiwhero Sluicing Company, of New Zealand, a form of apparatus for the saving of fine gold which is used in New Zealand beach deposits where the gold is accompanied by an excessive amount of black sand. A few of Mr. Young's remarks are here quoted:

> The modern fine gold washing plant, as used on the West Coast, consists of three main essential parts. The first is the hopper box with stone shoot, which receives the water and gravels from the tailrace connecting with the sluicing face, and separates the stones and shingles from the water and sands. The second comprises the "sand box" or "boil box," with its discharge ducts and other accessories, intermediate between the hopper and the tables. The third comprises the washing tables and their accessories. The three essential parts deal with the stuff from the face, and reduce it to concentrated gold and heavy sand ready for amalgamation.

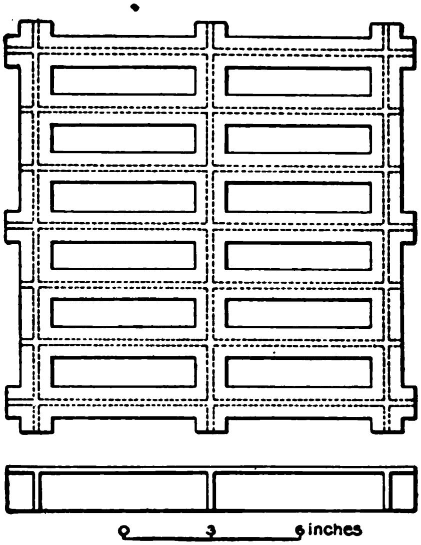

FIG. 48.—Iron-grate riffle, Seward Peninsula.

The introduction of a sand box or agitation box, in which the material is kept in agitation by being made to pass over and under a system of baffle boards before passing to the tables, as described by Mr. Young, is of obvious advantage in connection with the plant described. The principle will undoubtedly be of use in some of the washing plants of Alaska, as it provides a moderate stirring action without the use of mechanical power. The quantity of water economical for such a plant as the one described, with material screened to seven-sixteenths inch size on tables with a grade of 12 inches to 12 feet, is in the proportion of 40 miner's inches to each 10 feet of width of table. As at Oroville, the value of separate ducts from the feeding or sand box to each table has been proved.

The clearing of the gold from the accompanying minerals of high specific gravity is often difficult. In the creek workings on Bonanza

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[a] Report of the New Zealand Minister of Mines, 1902, p. 20.

Creek, Klondike, for example, of the total weight of gravel handled, 1 per cent of black sand, mostly magnetite, is caught with the gold in the clean-up. In American Creek, Alaska, the clean-ups are impeded by the presence of large quantities of barite pebbles. In the Fairbanks district red garnets and rutile, in some cases in quantity up to one-third of 1 per cent of the total material washed, are caught with the gold. In the Birch Creek district there is enough rutile in the auriferous sand to cause trouble in the clean-up. In Seward Peninsula the magnetite is in comparatively small amount. Garnets occur in the creek diggings up to 5 pounds to the cubic yard of gravel. In the beach and so-called "tundra" gold sands of the coastal plain from 3 per cent up to as high as 61 per cent of garnets occur.[a]

The use of quicksilver in the northern operations of America is limited. Even allowing for the extra expense and time consumed in the use and recovery of quicksilver, it is surprising that this important agency for saving fine gold is not more generally employed. It has been shown that the proportion of fine gold in the interior fields is large. The neglect to use quicksilver in attempting to save such gold can be considered only as a penny-wise-pound-foolish policy. It is needless to say that in all of the appliances above suggested for the saving of fine gold the use of quicksilver is imperative. Carelessness in the use of quicksilver, however, may result in increasing rather than diminishing the loss of fine gold. According to Bowie,[b] float quicksilver containing microscopic gold particles has been taken from the surface of the water 20 miles from the place where the amalgam entered the stream. A single flask of quicksilver is ample for the needs of the average creek operation of the Klondike or interior Alaska, and even with wasteful handling would last a season.

Whether quicksilver is used or not, it will be found advantageous to get the gold and amalgam as clean as possible on the floor of the sluices or tables before removing the valuable product. The comparatively small amount of heavy concentrates accompanying the creek gold of Seward Peninsula makes this possible in the frequent clean-ups of the tailrace. Skillful manipulation will accomplish the same result in the interior except where excessive quantities of magnetite occur. In one case on Bonanza Creek, Klondike, the fine concentrates which could not be removed with the magnetite after drying were skillfully separated by dry-panning.

Black sand or other concentrates have occasionally been found rich enough to pay for sacking and shipping to smelters after cleaning. Exaggerated reports of high assays in gold obtained from black sand frequently find credence. Nevertheless, whenever this concentrate accumulates in the sluices or tables in any considerable amount, it should be sampled and assayed to ascertain the amount of finely divided gold which still remains in it.

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[a] Brooks, A. H., Mendenhall, W. C., Collier, A. J., and Richardson, G. B., Reconnaissances in the Cape Nome and Norton Bay regions, Alaska, in 1900, U. S. Geol. Survey, 1901, p. 87.

[b] A Practical Treatise on Hydraulic Mining in California, 1885, p. 244.

## CLEANING, RETORTING, AND MELTING.

If amalgam is to be treated, it should be well stirred, either in buckets or large porcelain mortars, and the base material—sand, scraps of iron, etc.—which comes to the surface should be skimmed off.

This residue (which holds considerable amalgam) is concentrated by washing in pans or rockers, and the concentrations ground in iron (or porcelain) mortars and treated with more quicksilver. Any base material which floats on the surface of the bath is melted by itself to a base bullion. The remainder is added to the fine amalgam. The amalgam is strained from the quicksilver through drilling, and the dry amalgam is retorted in iron retorts.[a]

In cleaning the placer gold, when no quicksilver is used, a magnet inside a cotton sack passed through the dust will remove all the magnetite. Minerals of high specific gravity not attractable by the magnet are not easily removed by mechanical means. Rutile, garnets, and ilmenite are examples of these occurring in Alaska. In small operations the gold dust is roughly cleaned from these minerals by blowing and dry-panning. Where the amount of gold is considerable, a melting plant is advisable. In retorting amalgam, small hand retorts will in general be found adequate.

Before the amalgam is put in the retort the interior is coated with a thin wash of clay, which prevents the amalgam from adhering to the iron.

The amalgam should be carefully introduced and evenly spread. The iron pipe which connects the back end of the retort with the condenser must be clear of all obstructions, and under no circumstances should the amalgam be spread so that the pipe can possibly become choked, as in that case an explosion would probably ensue.

To avoid any danger arising from this source after the cover has been put on, lined with either clay or a mixture of clay and wood ashes, and securely clamped, the fire is lighted and the heat gradually raised, a dark-red heat being all that is necessary to thoroughly volatilize the quicksilver. Toward the end of the operation the heat is raised to a cherry-red color, at which it is kept until distillation ceases. The retort is allowed gradually to cool, and when cold is opened.[b]

A stream of cold water should be always flowing through the jacket which incloses the condensing pipe, so that by no possibility can mercurial vapor pass into the receiving vessel in which the lower end of the pipe terminates. The discharge end of the pipe should be kept under water during the retorting operation.

When gold accumulates in sufficient quantities to make the shipment of the dust inconvenient, the metal may be melted on the ground and molded into bricks. Both for convenience of shipment and facility in guarding against losses, this practice is to be recommended. A brief description of the essential features of this work follows.

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[a] Bowie, op. cit., p. 249.

[b] Bowie, op. cit.

In melting, a gasoline 2-jet furnace may be used. Gasoline under a pressure of 30 pounds enters a heating coil attached to the burner. The heating coil is so arranged and fed that liquid gasoline burning on the outside of the main feed vaporizes the gasoline which is used within the furnace. With the 30 pounds pressure used a very hot flame is the result. The gasoline supply should be stored without the building and brought in through pipes, so as to reduce the danger of explosion. An air barrel connected with the supply line and supplied with a small hand pump serves to keep the gasoline pressure as high as may be necessary. The furnace is made of sheet iron and lined with fire brick. A cover of asbestos and iron serves to retain heat. This should be so constructed as to allow the bolting on of new iron plates, as they burn off with the great heat generated. The operator should be supplied with asbestos gloves.

Crucibles are of various sizes. A No. 30 crucible, costing between $2 and $2.50, will hold from 900 to 1,000 ounces of impure gold or bullion with the necessary fluxing charge. They are composed of a mixture of fine clay and graphite, and to prevent breaking certain precautions should be observed. A new crucible should stand near the furnace, at least during two melts, and when first used should be heated gently. This precaution is not necessary when a crucible has once been used. After a crucible has been used five or six times it can not be depended on and it would be economy to discard it, as breaking during a melt is a source of great inconvenience and loss of time.

It is necessary to have an iron table near the furnace, upon which can be placed gold pans, shovels, tongs, etc.

Before the gold is placed in the crucible it should be accurately weighed and cleaned with a magnet, as above described.

In melting 900 to 1,000 ounces, three-fourths of a pound of borax should first be melted in the crucible as a flux. After the dust is poured in, one-fourth pound of soda with one-half pound of borax should be placed on top. The soda unites with the silica of the sand, but in perfectly clean gold is not needed. The borax unites with the iron occurring with the gold. When the dust is mixed with considerable iron pyrite it is well to add a small quantity of scrap iron. This, in uniting with the sulphur, forms iron sulphide, which comes off in the slag. If this precaution be not taken a hard matte, very difficult to remove, forms upon the brick.

During the melt it is necessary several times to skim the slag from the gold. A special instrument is used for this purpose, essentially a long rod bearing at its lower end an enlargement to which the slag will stick. After gathering a small quantity upon the skimmer it is brought out, and by rolling on the iron table is made into a smooth, disk-shaped mass upon the end of the rod. This operation is continued until the slag collected upon the skimmer becomes unwieldy, when it is cooled by plunging it in water and broken off. When the gold is reached it may be readily detected by its greater weight. A second flux of borax alone is then added, and when this has melted the product is ready for pouring.

A small crane to lift the crucible from the furnace and to pour the gold into the mold should be rigged overhead. Special tongs for lifting the crucible are made.

Though the mold should not be heated too much, it should always be raised to such a temperature that oil will burn on contact with it.

On removing the bullion brick from the mold it is placed in a pickle of three to four parts of water to one of nitric acid, which serves to clean the gold of surficial deposit. By using a hammer and a steel slag brush the brick is made ready for shipment.

## FINENESS OF GOLD.

The following table shows the fineness of Alaskan gold:

TABLE 13.—*Fineness of gold by districts and creeks.*

| District. | Fineness. | Authority. |
|---|---|---|
| **Juneau:** | *Per ounce.* | |
| Gold Creek | $17.50 | Operator. |
| Windfall Creek | 17.50 | Do. |
| Lemon Creek | 14.00 | Do. |
| **Porcupine:** | | |
| McKinley Creek | 16.80 | Members U. S. Geological Survey. |
| Porcupine Creek | 17.20 | Do. |
| **Nizina:** | | |
| Chittitu Creek | 18.71 | Operator. |
| Dan Creek | 18.87 | Do. |
| **Chisna:** | | |
| Miller Gulch | 18.41 | Do. |
| Slate Creek | 18.47 | Do. |
| Headwaters of Copper River | 17.80 | Do. |
| **Sunrise:** | | |
| Sixmile Creek | 17.34 | Members U. S. Geological Survey. |
| Crow Creek | 14.80 | Do. |
| **Atlin:** | | |
| Pine Creek | 16.43 | Operator. |
| Willow Creek | 15.65 | Do. |
| Spruce Creek | 16.50 | Do. |
| McKee Creek | 17.31 | Do. |
| Birch Creek | 16.23 | Do. |
| Gold Run | 16.00 | Do. |
| **Klondike:** | | |
| **Bonanza Creek:** | | |
| (*a*) High bench | 15.28 | Bank British North America. |
| (*b*) Creek | 16.43 | Do. |

TABLE 13.—*Fineness of gold by districts and creeks*—Continued.

| District. | Fineness. | Authority. |
|---|---|---|
| Klondike—Continued. | | |
| Hunker Creek: | *Per ounce.* | |
| (*a*) Bench | $17.75 | Bank British North America. |
| (*b*) Creek | 17.46 | Do. |
| Dominion Creek | 16.93 | Do. |
| Gold Run | 17.53 | Do. |
| Sulphur Creek | 16.48 | Do. |
| Bear Creek | 14.61 | Do. |
| Last Chance Creek: | | |
| (*a*) Bench | 14.12 | Do. |
| (*b*) Creek | 15.35 | Do. |
| Fortymile: | | |
| Wade Creek | 17.72 | Do. |
| Walker Fork | 18.03 | Operator. |
| Chicken Creek | 17.54 | Do. |
| Stone House Fork | 17.40 | Do. |
| Irene Gulch | 17.40 | Do. |
| Eagle: | | |
| American Creek | 17.61 | Bank British North America. |
| Mission Creek | 17.57 | Do. |
| Circle district: | | |
| Deadwood Creek | 16.73 | Do. |
| Mastodon Creek | 17.00 | Operator. |
| Mammoth Creek | 17.38 | Do. |
| Miller Creek | 17.25 | Do. |
| Eagle Creek | 18.93 | Bank British North America. |
| Mastodon Fork | 18.50 | Operator. |
| Woodchopper Creek | 18.32 | Do. |
| Fairbanks: | | |
| Fairbanks Creek | 17.76 | Bank British North America. |
| Cleary Creek | 17.24 | Do. |
| Pedro Creek | 18.40 | Operator. |
| Chatham Creek | 17.60 | Bank British North America. |
| Twin Creek | 18.40 | Operator. |
| Rampart: | | |
| Little Minook Creek | 19.24 | Bank British North America. |
| Little Minook, jr., Creek | 19.00 | Members U. S. Geological Survey. |
| Hunter Creek | 19.00 | Do. |
| Baker Creek | | |
| (*a*) Bench | 14.88 | Do. |
| Thanksgiving Creek | 15.17 | Do. |
| Glen Gulch | 16.00 | Operator. |
| Gold Run | 16.00 | Do. |

TABLE 13.—*Fineness of gold by districts and creeks*—Continued.

| District. | Fineness. | Authority. |
|---|---|---|
| Nome: | *Per ounce.* | |
| Anvil Creek | .......... | |
| (*a*) Bench | $18.60 | Bank of Cape Nome. |
| (*b*) Creek | 18.75 | Do. |
| Glacier Creek | 18.60 | Do. |
| Dexter Creek | 18.60 | Do. |
| Dry Creek | .......... | |
| (*a*) Creek | 18.39 | Operator. |
| (*b*) Bench | 17.30 | Do. |
| Beach diggings | 17.98 | Bank of Cape Nome. |
| High benches | 18.70 | Do. |
| Grass Gulch | 18.60 | Do. |
| Peluck Creek | 18.19 | Do. |
| Hastings Creek | 18.19 | Do. |
| Basin Creek | 18.00 | Operator. |
| Bonanza Gulch | 18.59 | Do. |
| Council: | | |
| Ophir Creek | 18.49 | Bank of Cape Nome. |
| Crooked Creek | 18.49 | Do. |
| Mystery Creek | 18.70 | Do. |
| Solomon: | | |
| Solomon River | 18.60 | Do. |
| Big Hurrah Creek | 18.39 | Do. |
| Port Clarence: | | |
| Gold Run Creek | .......... | |
| (*a*) Bench | 18.35 | Operator. |
| (*b*) Creek | 18.55 | Do. |
| Anikovik River | 19.00 | Do. |
| Sunset Creek | 17.50 | Do. |
| Fairhaven: | | |
| Bear Creek | 19.20 | Members U. S. Geological Survey. |
| Hannum Creek | 18.25 | Do. |
| Inmachuk River | 18.39 | Bank of Cape Nome. |
| Candle Creek | 17.77 | Do. |
| Kougarok: | | |
| Gold Run | 18.39 | Do. |
| Alder Creek | 18.39 | Do. |
| Kougarok River | 18.60 | Do. |
| Homestake Creek | 18.75 | Operator. |

## LABOR.

The following table (14) shows the average wages paid for labor in the mining districts of Alaska and adjacent Canadian territory during the summer of 1904. When the prices for winter work were available such have been added. The prevailing wages have been placed under the names of the districts as these are generally known, it being understood that the name of the nearest town, as, for example, Nome, stands for the creeks in the vicinity. Timbermen, carpenters, blacksmiths, and others whose pay is generally higher than that of ordinary laborers in the United States, are rarely paid more in Alaska. In districts where steam-thawing methods are employed engineers include pointmen. Hoistmen are not classed as skilled laborers, and where special prices for engineers obtain the men are usually in charge of large engines or pumps.

Two shifts are worked on most of the mines, especially in the summer. In hydraulic operations at night only one-third or one-fourth as many men are employed as on the day shift. In steam-thawing operations, where thawing is done at night, one or two pointmen and a fireman on the night shift generally take care of all the thawing, which will employ from 10 to 20 men in excavating, tramming, hoisting, and washing during the following day shift.

Ten-hour shifts are the rule throughout the northern placer fields. In large hydraulic operations the pipemen, as in other regions, generally work twelve hours. At only a few places were men seen working an eight-hour shift. At one of these, where men were shoveling into sluice boxes, it was proved by actual measurement of the ground that the number of yards shoveled per shift per man was greater than on adjacent claims where ten-hour shifts were in practice. In one instance on Ophir Creek, in Seward Peninsula, two eleven-hour shifts were worked, the men being paid 50 cents an hour and board. Experience has proved that the eight-hour shift for hard physical work is most economical in the operations in Alaska, where a large number of men are employed in shoveling and where every moment of the short season is valuable, and it would seem that the division of labor into three shifts of eight hours cheapens rather than increases the cost of handling material.

In some of the camps of Seward Peninsula a hospital fee of $2 per man per month is charged. This gives the laborer the services of a competent physician in case of sickness. Alaska is a singularly healthy country at all times of the year, and, although detailed statistics are not available, the proportion of deaths and illness to the total population would seem to be remarkably small.

TABLE 14.—*Scale of wages in Alaskan camps, including Dawson, Yukon Territory, and Atlin, British Columbia.*

| | Laborers. | Foremen. | Engineers. |
|---|---|---|---|
| Juneau | $3.50 to $4, without board; pipemen, $5. | $5, without board. | $5 a day, without board. |
| Chisna | $5 to $10 a day; some $12.50; some $90 to $150 per month. | | |
| Nizina: | | | |
| Copper Creek | $100 a month and board | | |
| Dan Creek | $7.50 a day | | |
| Sunrise | $4 a day, no board | | |
| Porcupine | $3 and board | $4 and board | |
| Atlin | $3.50 to $4.50 and board; $5 to $6.50, (pipemen), without board. | | $4 and board. |
| Dawson | $3.50 to $5 and board; winter, $3 and board. | $10 a day, no board. | $4 to $6 and board. |
| Fortymile | $5 and board; $8, no board. | | |
| Eagle | $5 and board | | |
| Rampart | $5 a day and board | | |
| Birch Creek: | | | |
| Deadwood Creek | $10 a day, no board | | |
| Birch Creek | $8 a day, no board; $6 to $8 a day, no board, is average. | | |
| Fairbanks | $10 a day, no board, general wages. | $14 a day, no board; some $6 and board; some $10 and board. | $12, no board; some $8 and board. |
| Some claims on Cleary Creek. | $5 and board | | |
| Some claims on Fairbanks Creek. | $6 and board | | |
| Nome | Summer, $5 a day and board; winter, $2.50 a day and board. | | $180 to $200 a month and board. |
| Council | Summer, $5 a day and board; winter, $3 a day with board. | $10 a day and board. | |
| Solomon | $5 and board | $7.50 and board. | |
| Kugruk | $6 a day and board | | |
| Fairhaven | $5 and board; some at $100 per month. | | |
| Port Clarence | $5 a day and board; $8 without board. | | |

## LUMBER AND FUEL.

### TIMBER.

The areal distribution of timber in Alaska does not by any means coincide with that of the placer gold. The South Coast region, in which there are extensive forests,[a] does not contain the most extensive alluvial gold deposits. The Juneau, Porcupine, and Sunrise districts are the only placer-gold fields developed within the area of heavy timber. In these three districts the large timber may be taken advantage of for local use. Under present conditions of transportation, however, this coastal timber forms no more of an asset in the consideration of the interior and the Seward Peninsula placer fields than as if it did not exist.

Spruce is commonly the timber on which the gold miner of the interior has to rely. It is very poor, both for fuel and for lumber, and in steam-producing quality a cord is the equivalent of not more than 700 pounds of Pennsylvania bituminous coal. The diameter of the spruce of the interior Yukon Valley rarely exceeds 18 inches, and averages 10 inches. The timber line varies from 2,000 to 3,000 feet above the sea, and many of the low-lying river valleys are fairly well forested. In the Tanana Valley and its tributaries near Fairbanks is found an exceptionally good growth of spruce timber, up to 18 inches in diameter, while birch, aspen, and cottonwood occur in less amount.

In the Fortymile district spruce trees up to 18 inches in diameter occur, but in general the growth does not differ from that found along the Yukon and Tanana bottoms.

At the head of the Tanana and White rivers the timber line reaches 3,000 feet, and there is a fairly good growth of spruce, birch, aspen, and cottonwood. The Kuskokwim and Koyukuk River valleys are also timbered with spruce.

The Sunrise and Chisna districts, which are nearer to the coastal strip than the regions mentioned above are better supplied with timber, spruce from 18 to 24 inches being common.

Those portions of Seward Peninsula which have attained importance from a placer-mining standpoint possess no timber except stunted willows. All mining operations must therefore depend on a foreign supply.

The prices for lumber in table 15 are for native or imported lumber, according to the timber resources of the district. In general all the interior camps make use of the native lumber. One of the difficulties experienced is to get good clear bottom boards for sluice boxes.

Sawmills are in operation at all the principal centers, Atlin, Dawson, White Horse, and Fairbanks. At Central House in the Birch Creek district a sawmill was formerly in operation, but has been abandoned. At Fairbanks, in the summer of 1904, two sawmills with a total daily capacity of 25,000 board feet were in operation.

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[a] Brooks, A. H., The geography and geology of Alaska: Prof. Paper U. S. Geol. Survey No. — (in preparation), forestry map, Pl. —.

Timber fires are frequent in the interior and are sometimes very destructive in dry seasons. There being no system of policing the forests for purposes of fire protection, the probability is that an increasingly large amount of timber will, as the population increases, be annually destroyed by fire.

## COAL.

Though there are extensive[a] coal fields in Alaska, practically none of them have been exploited. Much of the coal is of a lignitic character, but some very good bituminous coals have been found. In the South Coast province good coal occurs 20 miles from the coast near Controller Bay[b] and in the Matanuska basin[c] 50 miles from the head of Cook Inlet. Neither of these fields are yet within reach of transportation. Lignitic coals are present at many localities along the Pacific coast, but these, too, are undeveloped.

In the interior coal has been mined along the Yukon, but has not yet furnished a local fuel supply. The Yukon[d] coals include the lignite beds of the upper river and a better grade of bituminous coals which outcrop along the Yukon below Nulato. Lignite coal has also been found along the Koyukuk and on the Cantwell, a southern tributary of the Tanana. Though these coals may in the future furnish fuel for placer-mining operations they are an unknown factor.

Lignite has been found in the eastern part of Seward Peninsula, but its commercial value is unproved. A small mine has, however, been in operation in the Fairhaven district near Kotzebue Sound,[e] and has found a ready market for its output, which is of a lignitic character.

Near Cape Lisburne, 200 miles north of Nome,[f] bituminous coal occurs in commercial quantities, but has not yet been mined to any extent. It is still an open question whether it can compete with the imported coals.

The fuel question in Alaska may take a new phase with the introduction of gas-producing engines, for these lignites are found to be very effective power producers.

The following brief statement concerning recent tests with the gas producer at the United States Geological Survey coal-testing plant,

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[a] Brooks, A. H., The coal resources of Alaska: Twenty-second Ann. Rept. U. S. Geol. Survey, pt. 3, 1901, pp. 515-571.

[b] Martin, G. C., Petroleum fields of Alaska and Bering River coal fields: Bull. U. S. Geol. Survey No. 225, pp. 371-376.

[c] Stone, R. W., Coal resources of southwestern Alaska: Bull. U. S. Geol. Survey No. 259, pp. 151-171.

[d] Collier, A. J., The coal resources of the Yukon: Bull. U. S. Geol. Survey No. 218.

[e] Moffit, F. H., The Fairhaven gold placers, Seward Peninsula, Alaska: Bull. U. S. Geol. Survey No. 251, p. 67.

[f] Collier, A. J., Coal fields of the Cape Lisburne region: Bull. U. S. Geol. Survey No. 259, pp. 172-185.

St. Louis, has been courteously furnished for this report by Mr. Marius R. Campbell:

> The results of tests made during the past season at the United States Geological Survey coal-testing plant at the Louisiana Purchase Exposition at St. Louis proved conclusively that producer gas of such a quality that it may be used directly in a gas engine can be made from bituminous coals and lignites as well as from the highly carbonized fuels, such as anthracite, coke, charcoal, and wood, that alone have been used in the past for this purpose, and strange as it may appear, the lignites and lignitic coals have yielded better results than those ordinarily classed as bituminous. This means that the low-grade coals of the western half of the United States may have a value for the production of power equal to that of the best West Virginia coal as it is used at the present time.
>
> The equipment by which these results were obtained consists of a Taylor gas producer with economizer, scrubber, tar extractor, purifier, and gas holder, furnished by R. D. Wood & Co., of Philadelphia, and a gas engine of 235 B. H. P., furnished by the Westinghouse Machine Company, of Pittsburg. The great difficulty in the past in using bituminous coals in the producer has been the extraction of the tarry matter from the gas. With the above equipment the tar and sulphur in the gas is almost completely removed, so that these substances give little or no trouble in the gas engine.

It appears, then, that the interior must for the present depend for fuel on its timber resources, such as they are. The camps of Seward Peninsula will probably adopt crude oil as the most economical fuel if the price can be brought down to $2 a barrel delivered at Nome. There appears to be a need at Nome and some of the other large camps of an oil-tank and pipe-line system by which the oil can be cheaply transferred from steamers, stored, and pumped to the various producing creeks adjacent to the coast. The native spruce timber will afford a fuel supply to the region east of Niukluk River, if gold discoveries are made there.

For any of the camps in the South Coast region there is an abundant supply of fuel, as wood may be had for the cutting, and coal can be had at a comparatively low cost.

TABLE 15.—*Cost of firewood and flume lumber in Alaskan camps, including Dawson, Yukon Territory, and Atlin, British Columbia.*

| | Fuel. | Lumber per M, planed on one side unless otherwise stated. |
|---|---|---|
| Juneau district: | | |
| Windfall Creek | | $10.00 |
| Gold Creek | | 12.50-15.00 |
| Chisna district: | | |
| Headwaters of Copper River | | 150.00 |
| Slate Creek | | 150.00 |
| Daisy Creek | | 150.00 |
| Nizina district: | | |
| Chititu Creek | | 100.00 |
| Copper Creek | | 150.00 |
| Dan Creek | | 150.00 |
| Sunrise district: | | |
| Crow Creek | | 25.00-35.00 |
| Resurrection Creek | | 25.00 |
| Porcupine district: | | |
| Porcupine Creek | | 20.00 |
| McKinley and McCoon creeks | | 50.00 |
| Atlin district: | | |
| Pine Creek | | 40.00-50.00 |
| Boulder Creek | Spruce wood, $7 cord | |
| Spruce Creek | | 40.00-50.00 |
| McKee Creek | | 35.00-40.00 |
| Birch Creek | | 35.00-40.00 |
| Dawson district: | | |
| Bonanza Creek | Spruce wood, $15 cord | 70.00-90.00 |
| Eldorado Creek | | |
| Gold Run | Spruce wood, $9 cord | 70.00-100.00 |
| Sulphur Creek | Spruce wood, $9.50 cord | 80.00 |
| Upper Dominion Creek | Spruce wood, $11.50 cord | |
| Hunker Creek | Spruce wood, $11 cord | 60.00-90.00 |
| Fortymile district: | | |
| Wade Creek | | [a] 175.00 |
| Walker Fork | | [a] 100.00 |
| Chicken Creek | | 200.00-250.00 |
| Eagle district: | | |
| American Creek | | [a] 75.00 |
| Discovery Fork | | |
| Rampart district: | | |
| Gold Run Creek | | 120.00 |

[a] Rough.

TABLE 15.—*Cost of firewood and flume lumber in Alaskan camps, etc.*—Continued.

| | Fuel. | Lumber per M, planed on one side unless otherwise stated. |
|---|---|---|
| Circle district: | | |
| Deadwood Creek | Spruce wood, $12 cord | $100.00–125.00 |
| Independence Creek | | |
| Mastodon Creek | | 125.00 |
| Mammoth Creek | | |
| Eagle Creek | Spruce wood, $10 cord | 180.00 |
| Miller Creek | | |
| Woodchopper Creek | | 70.00 |
| Fairbanks district: | | |
| Fairbanks Creek | Spruce wood, $10 cord | 220.00–250.00 |
| Cleary Creek | | 225.00 |
| Chatham Creek | Spruce wood, $10 cord | 225.00 |
| Pedro Creek | Spruce wood, $7 cord | 100.00–150.00 |
| Gold Stream | Spruce wood, $7 cord | |
| Nome district: | | |
| Anvil Creek | Bituminous coal from $17 to $30 per ton.<br>Crude oil at Nome, $3 per barrel.<br>Gasoline distillate at Nome, $3.50 per 10-gallon case.<br>Wood, $12 cord on the Niukluk River. | 50.00 |
| Dexter Creek | | 50.00–60.00 |
| Glacier Creek | | 50.00–60.00 |
| Bourbon Creek | | |
| Dry Creek | | 45.00 |
| Peluk Creek | | 50.00 |
| Basin Creek | | 70.00 |
| Council district: | | |
| Ophir Creek | | 60.00–100.00 |
| Melsing Creek | | |
| Crooked Creek | | 125.00 |
| Solomon district: | | |
| Solomon River | | |
| Kassan Creek | | |
| Kougarok district: | | |
| Homestake Creek | | 150.00 |
| Fairhaven district: | | |
| Candle Creek | | 100.00–150.00 |
| Inmachuck River | | 70.00 |
| Kugruk River | | 150.00 |
| Port Clarence district: | | |
| Anikovik River | | 60.00 |
| Gold Run | | 125.00 |
| Buck Creek | | 50.00 |

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[a] Rough.

## ROADS AND ROAD BUILDING IN ALASKA.

### INTRODUCTION.

The means of getting from one part of the Territory to another and from the various towns and supply points to their tributary mining camps are very bad. The data concerning road construction, which were courteously furnished me by officers of the Canadian government based on Klondike experience, indicate that road building on an extensive scale is entirely feasible in the northern latitudes, and that roads can be constructed and maintained at moderate cost. In all probability Alaska is worse equipped with those improvements which contribute to progress and prosperity than any other area of like size lying in the domain of a civilized people.

Highway construction in Alaska is of the highest importance to the development of the Territory. The product of the gold-bearing gravels which have been already exploited is sufficient to support a considerable number of persons and to maintain towns of important size. The gold mining which has been done in the interior of Alaska has been conducted in spite of difficulties of transportation which would hardly be credible were they not substantiated by figures obtained in many parts of the Territory from responsible men.

It must not be assumed that the ordinary miner of Alaska is of a provident disposition. Many men of great energy, to whom is due the credit of having developed and opened large portions of this hostile country, are little disposed to take care for the future; yet they possess pluck and determination which is deserving of the highest praise. Outside of geologic and topographic exploration and making of excellent maps, and the furnishing of postal and telegraphic service, the miner, until the present session of Congress (1905), has been very little assisted.

An act of Congress was passed April 27, 1904, authorizing the appointment of road overseers and the creation of road districts in the Territory of Alaska, but this proving impracticable after a year's trial Congress has met the urgent demands for road building by the recent enactment of a statute which in two of its sections provides for the limited building and maintenance of wagon roads and trails.[a] Following are the first two sections of the said act:

*Be it enacted by the Senate and House of Representatives of the United States of America in Congress assembled,* That all moneys derived from and collected for liquor licenses, occupation, or trade licenses outside of the incorporated towns in the district of Alaska shall be deposited in the Treasury Department of the United States, there to remain as a separate and distinct fund, to be known as the "Alaska fund," and to be wholly devoted to the purposes hereinafter stated in the district of Alaska.

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[a] An act to provide for the construction and maintenance of roads, the establishment and maintenance of schools, and the care and support of insane persons, in the district of Alaska, and for other purposes.

One-fourth of said fund, or so much thereof as may be necessary, shall be devoted to the establishment and maintenance of public schools in said district; five per centum of said fund shall be devoted to the care and maintenance of insane persons in said district, or so much of said five per centum as may be needed; and all the residue of said fund shall be devoted to the construction and maintenance of wagon roads, bridges, and trails in said district.

SEC. 2. That there shall be a board of road commissioners in said district, to be composed of an engineer officer of the United States Army, to be detailed and appointed by the Secretary of War, and two other officers of that part of the army stationed in said district, and to be designated by the Secretary of War. The said engineer officer shall during the term of said detail and appointment abide in said district. The said board shall have the power and it shall be their duty, upon their own motion or upon petition, to locate, lay out, construct, and maintain wagon roads and pack trails from any point on the navigable waters of said district to any town, mining or other industrial camp or settlement, or between any such town, camps, or settlements therein, if in their judgment such roads or trails are needed and will be of permanent value for the development of the district; but no such road or trail shall be constructed to any town, camp, or settlement which is wholly transitory or of no substantial value or importance for mining, trade, agriculture, or manufacturing purposes. The said board shall prepare maps, plans, and specifications of every road or trail they may locate and lay out, and whenever more than five thousand dollars in the aggregate shall have been expended on the construction of any road or trail, contract for the work shall be let by them to the lowest responsible bidder, upon sealed bids, after due notice, under rules and regulations to be prescribed by the Secretary of War. The board may reject any bid if they deem the same unreasonably high, or if they find that there is a combination among bidders. In case no responsible and reasonable bid can be secured, then the work may be carried on with material and men procured and hired by the board. The engineer officer of the board shall in all cases supervise the work of construction and see that the same is properly performed. As soon as any road laid out by the board has been constructed and completed they shall examine the same and make a full, detailed report of the work done on the same to the Secretary of War, and in such report they shall state whether the road or trail has been completed conformable to the maps, plans, and specifications of the same. It shall be the duty of said board, as far as practicable, to keep in proper repair all roads and trails constructed under their supervision, and the same rules as to the manner in which the work of repair shall be done, whether by contract or otherwise, shall govern as in the case of the original construction of the road or trail. The cost and expenses of laying out, constructing, and repairing such roads and trails shall be paid by the Secretary of the Treasury out of the road and trail portion of said "Alaska fund" upon vouchers approved and certified by said board. The Secretary of the Treasury shall, at the end of each month, send by mail to each of the members of said board a statement of the amount available of said "Alaska fund" for the construction and repair of roads and trails, and no greater liability for construction or repair shall at any time be incurred by said board than the money available therefor at that time in said fund. The members of said board shall, in addition to their salaries, be entitled to receive their actual traveling expenses paid or incurred by them in the performance of their duties as members of said board.

It will be of interest to state that during the twelve months ending June 30, 1904, the total amount collected from license fees outside of incorporated towns was $145,043.65.[a]

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[a] MacLennan, M. F., Receipts and disbursements of the United States for the fiscal year ending June 30, 1904, Treasury Department, p.14.

As a little less than three-fourths of this amount is available for roads, it is evident that the work can not be pushed very rapidly.

It is probable that not more than 40 miles of standard road could be built and maintained in any one year in the central Yukon province, and not more than 20 miles of standard road in the South Coast province. Moreover, it is not likely that the fund will increase rapidly, as the number of incorporated towns is growing.

Though a beginning was made in the construction of a military horse trail from Valdez over Thompson Pass, the project was only partially successful on account of lack of funds for maintenance.

The recommendation of Mr. Alfred H. Brooks, of the Geological Survey, of an appropriation of $1,000,000 to be spent for wagon roads in Alaska is amply justified by the necessities of the case. It is probable that for this sum 900 miles of roads (300 of the Dawson standard wagon type and 600 for sleds) could be built in those parts of the country which would be most assisted by their construction. Provision should, however, be made for their annual maintenance. The inhabitants of Alaska would be as appreciative of such Federal aid as those of any portion of the American possessions, and, by such improvement in transportation facilities, the annual Alaskan product in gold would be greatly increased.

A serious detriment to the making of a road in Alaska is the thawing of the ground beneath the moss. It has been the universal experience that wherever the moss is cut into thawing immediately commences, and the trail which was passable becomes a filthy, slimy mass of mud, roots, and broken stones, a difficult route for men on foot, a slow and tiresome road for loaded animals, and an impassable obstacle to any sort of vehicle. In regions farther south, under temperate conditions, trails frequently are developed into fair wagon roads by much usage. Such developments can never take place in any part of the Northwest.

In the Northwest, where the ground is always subject to slight disturbances from alternate freezing and thawing, the roads can not be as durably constructed as in portions of the United States where similar topographic conditions prevail. The table of expenses furnished by the Canadian government for this report shows that the cost of maintenance does not exceed 15 per cent of the original mile-cost of the road. A feature that can not be too strongly impressed on those who have never seen the interior of Alaskan country is the extraordinary difference between the topography of the southeast coast, which is most often visited by tourists, and the portions of the country in which the rich placer deposits have been developed. In the coast region about Juneau, Admiralty and Baranoff islands, and in Prince William Sound (see Pl. XVI, *A*, p. 114) the needle-like peaks, precipitous slopes, cataracts in summer, avalanches in winter, and all the formidable conditions characterizing an alpine district make road building very costly. There is much rock work, and expensive bridges, trestles, and culverts are necessary. The cost of the wagon road from Juneau into Silver Bow basin was $3,750 a mile, and that of the wagon road built in 1898 from Skagway to the summit of the White Pass was $6,000 per mile. Compare this with the Yukon-Tanana gold district. In the vicinity of Fairbanks 13 miles of passable wagon trail, over what is known as the Ridge Trail going to the creeks, have been built at a cost of less than $100 per mile.[a] This road (see Pl. XXXVIII, *A*) was made through a stretch of birch and aspen timber; the trees were cut out and many of the stumps removed; no ditch was made along the sides and the road was wide enough for only one wagon. But it is passable, and freight rates have already been reduced from 25 and 30 cents per pound to 10 and 15 cents.

It may be objected that transportation sufficient to meet the needs of the miner can be accomplished in winter when the freight rates are very much cheaper. This is far from being the case. The winter transportation itself would be greatly assisted by good roads on which sleds of 4 and 5 ton capacity could travel. As to winter freighting, it may be stated that in the winter of 1903 a man contracted to transport a boiler weighing 5,600 pounds from the town of Fairbanks to Pedro Creek, a distance of 20 miles, for $2,000, and lost $300.

Furthermore, 75 per cent of the work of exploiting the gold-bearing gravels is carried on in the four months of the open season, and then the miners are in need of the bulk of their supplies. In summer the prices of commodities in the towns are frequently lower than in winter. The period during which supplies for the winter are generally brought in is from August 1 to September 1.

## COST AND METHOD OF CONSTRUCTING HIGHWAYS.

Considered from a highway standpoint, Alaska, owing to its varying topography and natural conditions, may be divided into three provinces—the South Coast province, the central Yukon province, and the Seward Peninsula province. Opportunity was had to inspect the excellent wagon roads which have been built in the northwest portion of Canada by the Canadian government. The figures courteously given by the officers of the Canadian department of public works relative to the cost of highway construction in the Yukon Territory are considered representative of the probable cost of such construction in the central Yukon province of Alaska. This and other information is embodied in table 16, and portions of it are elaborated in the succeeding paragraphs. The object of the table is to show, in a form easy for comparison, the costs of road building in the Northwest as exemplified by and estimated from previous experience.

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[a] So low a cost would not have been possible had not teamsters hauled all necessary supplies gratis.

*A.* ROAD IN FAIRBANKS DISTRICT.

*B.* ROAD IN ATLIN DISTRICT, BRITISH COLUMBIA.

The portions of Alaska represented by Alaska Peninsula and neighboring islands, and the Copper River basin, north of Prince William Sound, have not been especially considered, as there appears to be no present need there of expensive routes of communication. In general it may be said, however, that the cost of construction in Alaska Peninsula is the same as in Seward Peninsula, and the cost in the Copper River basin the same as in the central Yukon province.

TABLE 16.—*Cost of road construction.*

| Locality. | Authority. | Construction, cost per mile. | Annual maintenance, cost per mile. | Notes. |
|---|---|---|---|---|
| South Coast province: | | | | |
| Juneau | Private constructors. | $3,750 | .......... | Good road built under great topographic difficulties. |
| Skagway—White Pass. | ....do | 6,000 | .......... | Road no longer used. |
| British Columbia: | | | | |
| Atlin | B. C. Atlin, gold commissioner. | 300–1,000 | .......... | Good roads easily built, very little corduroy. |
| Yukon Territory: | | | | |
| Dawson and vicinity. | Canadian department of public works of Yukon Territory. | 1,500–3,300 | $350 | Standard highways constructed as described below. |
| Do | ....do | 250–350 | 25 | Sled or winter trails; rarely used in summer. |
| Seward Peninsula: | | | | |
| Nome and vicinity. | Estimates based on cost of railways already constructed. | 3,000–4,000 | 500 | Standard highways constructed as described below. |
| Do | Private railway enterprises now in operation. | 6,000 | .......... | Railway — 3-foot gauge, 20-pound rail, 2,600 ties to mile. |
| Central Yukon: | | | | |
| Tanana, Birch Creek, and Koyukuk districts. | Estimate based on work done by Canadian government in Yukon Territory. | 1,200–3,000 | 350 | Standard highways constructed as described below. |
| Do | ....do | 300 | 25 | Sled or winter trails. |

## SOUTH COAST PROVINCE.

Wagon roads, while extremely useful in this portion of Alaska, cost so much that it is not likely they will be built except as heretofore, by private enterprise, to accommodate individual mining properties or groups of mines. The province is likely to prove in the future, as it has proved in the past, more important from the standpoint of quartz mining than of alluvial mining. The construction of roads is none the less desirable on this account, and should any large area capable of producing mineral be found at a point remote from the coast, a highway to it over which freight could be handled all the year round would be highly desirable. Winter conditions are exceedingly severe in the higher mountains. In the construction of the White Pass and Yukon Railway, 110½ miles in length, 1,500 men were employed for twenty-six months. Train service is maintained throughout the winter under considerable difficulty, both heavy winds and heavily drifted snow being frequently encountered. Rotary snow plows have to be run daily, encountering drifts from 15 to 35 feet in depth. The average cost of handling snow for a season is $75,000. Characteristic South Coast province topography is illustrated by Pl. XVI, *A*, p. 114.

## ATLIN DISTRICT AND YUKON TERRITORY.

In the Atlin district of British Columbia, which was visited during the present investigation, the topography is comparatively favorable to road construction, and it will be noticed from the table that the cost is low. Pl. XXXVIII, *B*, p. 220, shows a characteristic view of an Atlin road.

The Yukon Territory of Canada offers the most instructive lesson in practical highway construction in the north, and affords a striking contrast in this respect to the adjacent American domain. The officers of the department of public works of the Yukon Territory, notably Mr. S. A. D. Bertrand, the superintendent, and Mr. W. Thibaudeau, the Territorial engineer, were kind enough to place at my disposal the data relative to the building and maintenance of the public roads. The following summary can not fail to be of interest in this connection:

The longest road in the Territory is from Dawson to White Pass, 342 miles. Its history dates from 1899.

Prior to 1899 all freight from Dawson was transported to the mining creeks by pack animals in summer, and sledded either with dogs or horses over the ice and snow in winter.

The following table shows the wagon roads and sled roads constructed:

*Roads constructed from 1899 to 1903.*

| | 1899. | 1900. | 1901. | 1902. | 1903. | Total. |
|---|---|---|---|---|---|---|
| | *Miles.* | *Miles.* | *Miles.* | *Miles.* | *Miles.* | *Miles.* |
| Wagon roads | 45 | 32 | 63.00 | 85.81 | 26 | 251.81 |
| Sled roads | 160 | 10 | 80.25 | 372.00 | 4 | 626.25 |

*Amount expended from 1899 to 1903 in constructing and maintaining roads and bridges in Yukon Territory, Canada.*

| | |
|---|---|
| 1899 | $50,000.00 |
| 1900 | 205,609.72 |
| 1901 | 147,174.90 |
| 1902 | 228,543.88 |
| 1903 | 398,789.95 |
| Total | 1,030,118.45 |

Average cost of construction for wagon road was $1,500 to $3,300 per mile. Average cost of construction for sled or winter trail was $250 to $350 per mile. The cost of repair and maintenance of wagon roads was $350 per mile per annum. The cost of repair and maintenance of sled roads was $25 per mile per annum.

The following comparative statement shows transportation charges, per 100 pounds between Dawson and a few of the important mining centers in 1899, previous to the construction of the wagon roads, and in 1904. The improved condition means a direct benefit to the miners of at least 600 per cent on all supplies and machinery freighted by wagons over first-class roads:

*Transportation charges, per 100 pounds, from Dawson to important mining centers in Yukon Territory, Canada.*

| From Dawson to— | Distance. | 1899. | 1904. |
|---|---|---|---|
| | *Miles.* | | |
| Grand Forks | 12 | $7.00 | $1.00 |
| Gold Bottom | 20 | 8.00 | 1.50 |
| Caribou | 33 | 12.50 | 2.00 |
| Discovery Sulphur | 35 | 12.50 | 2.00 |
| Gold Run | 55 | 18.00 | 3.00 |

The following is a statement of the localities served by wagon roads:

*Localities served by wagon roads, with mileage, in Yukon Territory, Canada.*

| | Miles. |
|---|---|
| Bonanza Creek | 18.00 |
| Lovett Gulch | 1.50 |
| Adams Gulch | 1.75 |
| Eldorado Creek | 7.65 |
| Calder Creek | 5.60 |
| Gold Hill | .50 |
| Quartz Creek | 5.00 |
| Indian River and Eureka Creek | 18.00 |
| Hunker Creek | 24.35 |
| Bear Creek | 3.00 |
| Last Chance Creek | 2.50 |
| Gold Bottom Creek | 5.40 |
| Dominion Creek | 16.25 |
| Sulphur Creek | 9.56 |
| West Dawson | .41 |
| Henderson Creek | 13.00 |
| Duncan Creek (Stewart River district) | 20.00 |
| Livingstons Creek (Big Salmon district) | 16.00 |
| Copper King mines (White Horse district) | 4.20 |
| Grafter copper mine | 5.44 |
| Ridge road and spurs to Grand Forks, Gold Bottom, Caribou, and Gold Run | 73.70 |
| Total | 251.81 |

Following is a statement of the localities served by sled roads:

*Localities served by sled roads, with mileage, in Yukon Territory, Canada.*

| | Miles. |
|---|---|
| Old River cut-off from Lake Labarge to Hutchi, cut out in 1899 | 160.00 |
| Dawson-White Horse, overland mail route | 350.00 |
| Glacier-Miller Creek, Sixtymile district | 58.25 |
| Bucher Creek, Sixtymile district | 20.00 |
| Lepine Creek, quartz mines | 14.00 |
| Lower Sulphur | 8.00 |
| Last Chance (upper end) | 3.00 |
| Lower Dominion | 6.00 |
| Cut-offs, Stewart River | 20.00 |
| Duncan Creek (upper part) | 4.00 |
| Total | 643.25 |
| Less Eureka end of Dawson-White Horse trail (covered into wagon road) | 17.00 |
| | 626.25 |

The estimate for the amount required for maintenance of roads and bridges in the Yukon Territory for the year ending June 30, 1904, is $103,190.

The men employed in road construction are paid $5 and board per day of ten hours. A team of horses, used in scraping or carting, is reckoned at $25 per day, including wages of the driver. Foremen are paid $6 and board per day of ten hours.

*A*. OGILVIE BRIDGE, KLONDIKE RIVER.

*B*. SIX-MULE TEAM DRAWING 8 000 POUNDS, KLONDIKE.

Several bridges of a permanent character have been built in the Territory, the principal of which is the Ogilvie steel bridge over Klondike River, the cost of which was $35,000. It is seen in Pl. XXXIX, *A*.

The method of constructing the road in the Yukon Territory, where frozen muck and gravel flats are traversed, is illustrated in fig. 49. This section was drawn from information furnished by Mr. W. Thibaudeau, Territorial engineer of the Yukon Territory. It has been the experience that over frozen ground this type of road, costing $3,200 per mile, can be maintained with much less expense than roads running along the hillsides. With roads of the kind here illustrated it has been found best to leave the moss intact under the bed of poles, for it protects the ground from thawing; the black frozen muck, having the consistency of solid stone, remains as a firm bed. The inside faces of the side ditches are not always banked with sod, but such banking affords additional protection. The ditches are cut either entirely in muck or partly in muck and partly in underlying gravel. These conditions vary with the thickness of the muck.

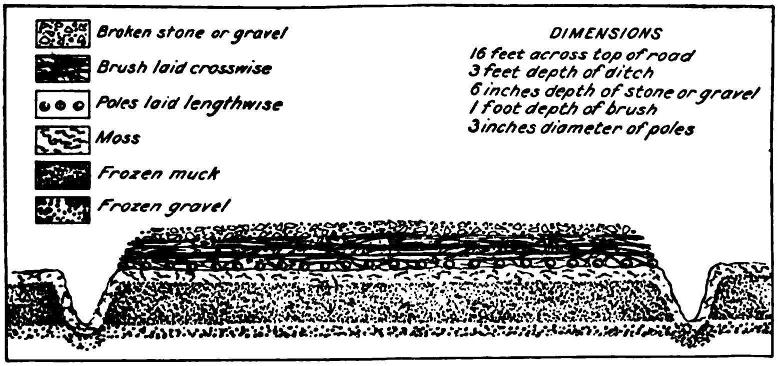

FIG. 49.—Method of constructing road over frozen muck, Klondike.

On the other hand, when the moss blanket is cut into, as is necessarily the case in all side-hill cutting, the sun and the seepage water gradually thaw the frozen muck, which becomes a soft, slimy mass. The cost of maintenance of such roads has been found so much greater than that of roads on flat ground that the latter are now constructed, even if the distance between terminal points is greater.

Pl. XXXIX, *B*, shows a 6-mule team hauling 8,000 pounds with a trailer on an upgrade stretch of road in Hunker Creek, near Dawson. Pl. XL, *A* (p. 226), from a photograph taken in August, 1904, shows a 6-horse team endeavoring to haul a load of 3,300 pounds from Fairbanks, on Tanana River, to Cleary Creek, in the Fairbanks district of Alaska. Not half a mile from the point where this picture was taken, on the road to Cleary Creek, a cart bearing a 15-horsepower boiler was seen broken down, its wheels buried in mud and water.

Two miles from the town of White Horse, Yukon Territory, on the road to the copper deposits, is an excellent stretch of corduroy road extending over a marshy piece of unfrozen ground. The 6-inch timbers used for lagging have all been flattened on the upper side for a width of 15 feet in the central portion, so that a rough wagon track is formed. Thus a much evener wear on the lagging is maintained than would be the case if the timbers were left in their original round condition.

## CENTRAL YUKON PROVINCE.

Mr. Alfred H. Brooks, in his article entitled "Placer Mining in Alaska in 1903,"[a] has suggested the following routes for Government roads in the central Yukon province of Alaska:

> (1) A road from Eagle to the Tanana, the Chistochina, and to Valdez, on the coast, a distance of approximately 400 miles. This probably should follow the present well-established trail, which is used as winter and summer mail route. The most important part of the road would be that between Eagle and the camps of the Fortymile region. An alternate route would be to Fairbanks, on the Tanana, from Copper River by way of the Delta River valley, a distance of about 300 miles. (2) A road to extend from Circle, on the Yukon, through the Birch Creek and Fairbanks district to the Tanana, a distance of about 150 miles. (3) Rampart, on the Yukon, to be connected with the mouth of Baker Creek, on the Tanana, by a road which would open up the Minook and Baker districts, a distance of less than 50 miles. (4) A hundred miles or so of road to be built in the Koyukuk region to connect the gold-bearing creeks with the head of steamboat navigation on Koyukuk River. It is believed that these roads would form a system of main arteries by which most of the placer fields could easily be reached, and that the production of the mines would thereby be so much increased as to fully justify the expense.

Well-constructed roads following the above routes would result in making freight rates in the placer districts which the roads would traverse one-fourth less than at present. The operations in the Birch Creek and New Fairbanks placer districts, for example, are directly dependent on prices of supplies laid down on the claims. The pay streaks in the broad creek valleys frequently become wider and wider as freight rates become lower. Such a statement would appear absurd if applied to regions where the charge for freighting represents only a small percentage of the original cost of the commodities. Where, however, the cost of material, especially machinery, is rendered double or more by freight charges, transportation facilities become a highly important factor in considering whether a given piece of ground is or is not workable. Prices for mining supplies in the town of Fairbanks, the supply point for the new gold-placer region on tributaries of Tanana River, are by no means cheap. As compared with prices on the claims which lie from 15 to 25 miles inland, however, they are

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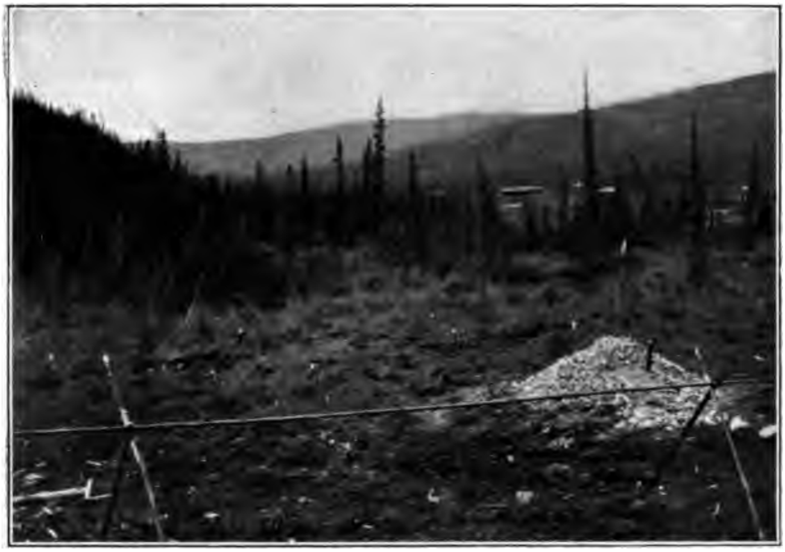
[a] Bull. U. S. Geol. Survey No. 225, 1904, p. 57.

*A*. SIX-HORSE TEAM DRAWING 3,300 POUNDS, CLEARY CREEK, FAIRBANKS.



*B*. CREEK TOPOGRAPHY, FAIRBANKS DISTRICT.

strikingly low. For example, a hundred feet of 8-inch 16-gage hydraulic riveted steel pipe costs in Fairbanks $175. On Fairbanks Creek, 20 miles away, the same 100 feet of pipe, with freight at 20 cents per pound, costs, if transported in summer, $301, representing a freight charge of $126. In the Klondike, where the topography is nearly the same, the same pipe would be landed on a claim 20 miles from Dawson for a freight charge of $9.45.

Anything worse than the present condition of the trails in the interior of Alaska is difficult to conceive. Men ignorant of the use of a map and compass are frequently lost in the vast maze of rolling mountains and undergo hardship and starvation. It is a curious fact, dependent on the peculiar frozen conditions, that the great river flats, where the present trails are at their worst, are the very places where wagon roads of the Dawson type can be most easily maintained. As for the hill country, the roads can, in many cases, be led almost continuously for distances of from 5 to 20 miles along the crests of gently rolling divides.

Pl. XL, *B*, shows one of the important producing creeks of central Alaska, where a road is badly needed, and Pl. XLI, *A*, illustrates the present mode of transportation in that district.

### SEWARD PENINSULA PROVINCE.

In the central and western portions of Seward Peninsula, namely, the Nome, Solomon, Council, Topkok, Kougarok, Teller, and York districts, any projects of installation of roads must take into account the entire absence of native timber. The region so defined is the one which has attained importance as a gold-producing area, while the eastern portion of the province, where scant growth of spruce does occur, is not at present productive. Pl. XLII illustrates typical Seward Peninsula topography.

Owing to these conditions wagon-road construction which employs the use of timber in any form will be expensive, and as seen by the table given previously in this paper, the cost of building the narrow-gage railways which have done excellent service in Seward Peninsula is $6,000 per mile, exclusive of rolling stock, while a wagon road of the Dawson type will rarely fall below $4,000 per mile. The rates for hauling on these railways may be reckoned at $1 per ton per mile. In summer, wagons with broad tires now haul freight over roads which have, as it were, made themselves, at rates of $1.50 to $3 per ton per mile, varying according to the nature of the country traversed.

It is questionable whether wagon roads, extensively built and aiming at permanency, will ever be constructed in Seward Peninsula to supply the needs of the gold-mining communities. The Nome

Arctic Railway, from Nome to Anvil Creek, 12 miles long, the ties of which are laid for a portion of its length on a 2-inch plank bed, appears to serve a useful purpose. Only seventy days were consumed in the construction of the 12 miles. The Wild Goose Railway, also of 3-foot gage, from Council to Ophir Creek, has been constructed in a similar manner. Sixteen miles of a standard-gage railway, the Council City and Solomon River Railway, have also been completed. Considering the comparative cheapness with which railway material can be hauled at various points in Seward Peninsula, it is likely that short lines of railway will continue to be built by private enterprise to supply the several inland mining communities.

The suggestion is here made that narrow-gage pole tramways laid over the tundra might in some cases be used in parts of Seward Peninsula for long hauls. They would be expensive to construct and maintain, but would not, of course, compete with winter transportation.

In view of the small cost of narrow-gage railways in Seward Peninsula and the small annual maintenance cost, as compared with what it would probably cost to maintain wagon roads, the Federal Government should aid in tramway rather than wagon-road construction in that portion of Alaska. The light industrial railways now manufactured by various firms could be constructed with special modifications for the tundra country, and could be shipped, knocked down, to different portions of the peninsula.

In conclusion, I would say that of the three provinces of Alaska above specified, the central Yukon province is most in need of Federal aid for highway construction. By this means placer districts now lying idle could be made to support a numerous population, and it is doubtful whether any other form of assistance would prove as beneficial as stimulating prospecting for new placer fields.

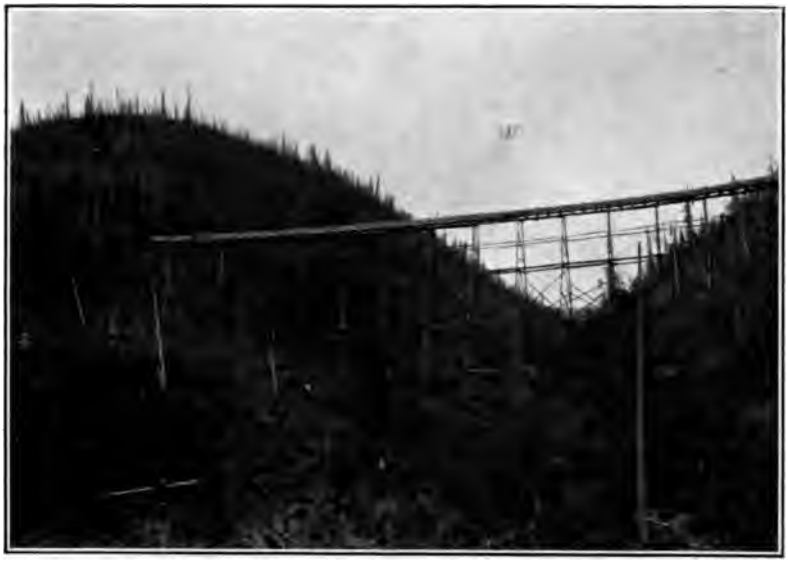
## FREIGHT RATES.

### FREIGHT RATES FROM THE STATES.

The present freight and passenger rates from Tacoma, Seattle, and San Francisco to the main supply points of Alaska accessible by ocean-going steamers are given in table 17 (pp. 230-231).

Freight rates are fairly constant from year to year, but passenger rates vary from one season to another, according as the different steamship companies are competing or temporarily combined. Return passenger rates may be reckoned at generally 25 per cent higher than north bound, but occasionally rise to 100 per cent higher. During 1904 the return rates from Nome were double those outgoing. It

A. PACK TRAIN, BIRCH CREEK DISTRICT.



B. TRESTLED FLUME IN EAGLE DISTRICT.

should be noted also that the return rates are higher on the last steamers than on those returning early in the season.

Service is maintained all the year round, with the exception of the Seward Peninsula service, St. Michael, and points far up Cook Inlet. Between these points and Seattle, service can be reckoned on from May 15 to October 15 only.

TABLE 17.—*Ocean freight and passenger rates for 1904 from*

| | | Loose machinery and iron articles in pieces weighing less than 2,000 pounds. | Machinery, single pieces weighing 2,000–4,000 pounds. | Machinery, single pieces weighing 4,000–6,000 pounds. | Machinery, single pieces weighing 6,000–8,000 pounds. | Machinery, single pieces weighing 8,000–10,000 pounds. | Machinery, single pieces over 10,000 pounds. |
|---|---|---|---|---|---|---|---|
| | | *Per ton.* | *Per ton.* | *Per ton.* | *Per ton.* | *Per ton.* | |
| | Wrangell | $9.00 | $11.00 | $14.00 | $18.00 | $22.00 | Special contract. |
| | Juneau | 10.00 | 12.00 | 15.00 | 19.00 | 23.00 | ....do ... |
| | Skagway | 11.00 | 13.00 | 16.00 | 20.00 | 24.00 | ....do ... |
| | Yakutat | 13.00 | 15.00 | 18.00 | 22.00 | 26.00 | ....do ... |
| | Valdez | 11.00 | 13.00 | 16.00 | 20.00 | 24.00 | ....do ... |
| | Seward | 16.00 | 18.00 | 21.00 | 25.00 | 29.00 | ....do ... |
| | Hope and Sunrise | 19.50 | 21.50 | 24.50 | 28.50 | 32.50 | ....do ... |
| | Sand Point | 18.00 | 20.00 | 22.50 | 27.50 | 32.50 | ....do ... |
| Service June 1–Oct. 1 only. | St. Michael | 15.00 | 15.00 | 20.00 | 20.00 | 35.00 | $50 and higher. |
| | Cheenik | 15.00 | 15.00 | 20.00 | 25.00 | 35.00 | ....do ... |
| | Solomon | 15.00 | 15.00 | 20.00 | 25.00 | 35.00 | ....do ... |
| | Nome | 15.00 | 15.00 | 20.00 | 25.00 | 35.00 | ....do ... |
| | Teller | 25.00 | 25.00 | 30.00 | 35.00 | 45.00 | $60 and higher. |
| | York | 25.00 | 25.00 | 30.00 | 35.00 | 45.00 | ....do ... |
| | Bluff | 25.00 | 25.00 | 30.00 | 35.00 | 45.00 | ....do ... |
| | Kiwalik | 45.00 | 45.00 | .......... | .......... | .......... | .......... |

Coal is shipped from different points to Alaska, and the local price is given in tables for different Alaska towns.

Coal in sacks per ton, 2,240 pounds, Seattle to St. Michael or Nome, $12.

Freight and passenger rates from San Francisco to any of the above points do not exceed 5 per cent above Seattle rates.

The winter passenger trip between Seattle and Nome, via White Pass and Dawson, includes steamer, rail, horse-stage, and dog-sled travel, consumes three months, and costs about $1,000.

*Seattle to Alaska coast points, including lightering when necessary.*

| General merchandise, including boxed machinery. | Boats and canoes. | Lumber. | Powder, dynamite and black. | Hay, double compress. | Grain. | Horses, mules, cattle. | Dogs. | Passengers. | | | Return freights. | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | First-class. | Second-class. | Steerage. | Ore and concentrates not exceeding $100 per ton value. 10 tons or under. | 10 to 110 tons. | 100 to 300 tons. | Bullion and dust (per $1,000). |
| *Per ton.* | | *Per M ft.* | *Per ton.* | *Per ton.* | *Per ton.* | *Per head.* | *Per head.* | | | | *Per ton.* | *Per ton.* | *Per ton.* | |
| $8.00 | $10.00–25.00 | $9.00 | $24.20 | $11.00 | $6.20 | $30.00 | $5.00 | $20.00 | $15.00 | ........ | $4.00 | $3.50 | $3.00 | $7.50–10.00 |
| 9.00 | 10.00–25.00 | 10.00 | 25.20 | 12.00 | 7.20 | 30.00 | 5.00 | 25.00 | 16.00 | ........ | 4.00 | 3.50 | 3.00 | 7.50–10.00 |
| 10.00 | 10.00–25.00 | 11.00 | 26.20 | 13.00 | 8.20 | 30.00 | 5.00 | 30.00 | 20.00 | ........ | 4.00 | 3.50 | 3.00 | 7.50–10.00 |
| 12.00 | 10.00–25.00 | 15.00 | 37.00 | 15.00 | 12.00 | 30.00 | 5.00 | 32.00 | 21.00 | ........ | 4.00 | 3.50 | 3.00 | 7.50–10.00 |
| 10.00 | 10.00–25.00 | 13.00 | 35.00 | 13.00 | 16.00 | 30.00 | 5.00 | 40.00 | 25.00 | ........ | 4.00 | 3.50 | 3.00 | 7.50–10.00 |
| 15.00 | 10.00–25.00 | 17.50 | 40.00 | 18.00 | 15.00 | 30.00 | 5.00 | 45.00 | 31.00 | ........ | 4.00 | 3.50 | 3.00 | 7.50–10.00 |
| 18.50 | 19.00–34.00 | 14.50 | 43.50 | 21.50 | 18.50 | 40.00 | 6.50 | 57.00 | 42.00 | ........ | ...... | ...... | ...... | 7.50–10.00 |
| 17.50 | 12.50–27.50 | 10.00 | 40.00 | 18.00 | 17.50 | 35.00 | 7.50 | 70.00 | 50.00 | ........ | 4.50 | 3.50 | 3.50 | 7.50–10.00 |
| | | 22.00 | | 20.00 | 18.00 | | | | | | | | | |
| 15.00 | Freight rates from Seattle or San Francisco vary from $10 to $15 per ton of 40 cubic feet. There is no classification. | | | | | 60.00 | ...... | 50.00–100.00 | ...... | $25.00–60.00 | $5.00 | | | 7.50–10.00 |
| 15.00 | | | | | | 60.00 | ...... | 50.00–100.00 | ...... | 25.00–60.00 | 5.00 | | | 7.50–10.00 |
| 15.00 | | | | | | 60.00 | ...... | 50.00–100.00 | ...... | 25.00–60.00 | 5.00 | | | 7.50–10.00 |
| 15.00 | | | | | | 60.00 | ...... | 50.00–100.00 | ...... | 25.00–60.00 | 5.00 | | | 7.50–10.00 |
| 25.00 | | | | | | 60.00 | ...... | 60.00–110.00 | ...... | 30.00–65.00 | 6.00 | | | 7.50–10.00 |
| 25.00 | | | | | | 70.00 | ...... | 70.00–120.00 | ...... | 35.00–70.00 | 6.00 | | | 7.50–10.00 |
| 25.00 | | | | | | 60.00 | ...... | 60.00–110.00 | ...... | 35.00–70.00 | 5.00 | | | 7.50–10.00 |
| 45.00 | .............................. | | | | | 85.00 | ...... | 80.00–130.00 | ...... | 55.00–90.00 | 10.00 | | | 7.50–10.00 |

In table 18 are given the through freight and passenger rates from San Francisco or Seattle to Dawson, in Yukon Territory, and to Eagle, Circle, Fort Gibbon, and Fairbanks, in Alaska, via the White Pass route.

Table 19 gives the through rates from San Francisco or Seattle to the above-named points, via St. Michael and Yukon and Tanana rivers.

While during the open or summer season the all-water transport of freight via St. Michael is cheaper, the river steamboat service has not yet been developed to a sufficient extent to insure prompt and regular delivery. Should the Tanana gold fields prove important, however, the all-water route will probably be the one most practical for the transportation of supplies. It has the additional recommendation that by means of it the crossing and recrossing of the Canadian frontier is avoided. On the other hand, via the White Pass route, freight can be landed at Fairbanks much earlier than via the all-water St. Michael route.

It should be stated, in the consideration of table 18, that the White Pass and Yukon Railway has established three classifications of freight for through shipments, the regular rates for which, on less than 10 shipments from Seattle to Dawson for the summer season, are as follows:

Class A, including mining tools and articles bulky and not easily destructible, $65 per ton.

Class B, including more fragile articles, $80 per ton.

Class C, including small supplies, $95 per ton.

Shippers should remember that a package containing any article coming under a higher class than that comprising the bulk of the package is charged at the highest rate. This applies also to unspecified packages.

A list of mining supplies, on which a special reduction is made for transport from Seattle to Dawson between July 1 and August 15, is given following table 18.

The winter freight rates, via the White Pass route, exceeding $560 per ton from Seattle to Dawson need not be considered, as they are prohibitive, except in emergency cases. This route necessitates a haulage of nearly 300 miles by sleds.

Table 18.—*Through freight and passenger rates for 1904 from Seattle to interior Yukon points, via White Pass route, rail and water—season June 15 to September 15.*

| Seattle to— | Loose machinery and tools in pieces less than 2,000 pounds.[a] | Machinery, single pieces weighing 2,000-4,000 pounds. | Machinery, single pieces weighing 4,000-6,000 pounds. | Machinery, single pieces weighing 6,000-8,000 pounds. | Machinery, single pieces weighing 8,000-10,000 pounds. | Machinery, single pieces weighing over 10,000 pounds. | General merchandise, including boxed machinery.[a] | Lumber, rough and flume.[a] | Powder, dynamite, and black. | Hay, double compressed.[a] | Grain.[a] | Horses and cattle. | Passengers.[b] | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | | | | First-class. | Second-class. |
| | *Per ton.* | *Per ton.* | *Per ton.* | *Per ton.* | *Per ton.* | *Per ton.* | | *Per M ft.* | | | | *Per head.* | | |
| Dawson | $65.00 | $65.00 | $80.00 | $80.00 | $95.00 | $100-150.00 | $80.00 | $97.00 | $120.00 | $65.00 | $65.00 | $80.00 | $80.00 | $65.00 |
| Fortymile | 75.00-80.00 | 83.00 | 100.00 | Special contract. | Special contract. | Special contract. | 90.00 | 112.00 | 135.00 | 75.00 | 75.00 | 90.00-95.00 | 87.00 | 70.00 |
| Eagle | 77.00-82.00 | 90.00 | 104.00 | do | do | do | 92.00 | 115.00 | 138.00 | 77.00 | 77.00 | 92.00-98.00 | 89.00 | 72.00 |
| Circle | 83.00-90.00 | 98.00 | 116.00 | do | do | do | 98.00 | 124.00 | 147.00 | 83.00 | 83.00 | 98.00-107.00 | 97.00 | 78.00 |
| Fort Yukon | 86.00-95.00 | 104.00 | 122.00 | do | do | do | 101.00 | 129.00 | 151.00 | 86.00 | 86.00 | 101.00-111.00 | 101.00 | 81.00 |
| Rampart | 91.00-102.00 | 111.00 | 132.00 | do | do | do | 106.00 | 136.00 | 159.00 | 91.00 | 91.00 | 106.00-119.00 | 111.00 | 88.00 |
| Fort Gibbon | 93.00-104.00 | 115.00 | 136.00 | do | do | do | 108.00 | 139.00 | 162.00 | 93.00 | 93.00 | 108.00-122.00 | 114.00 | 90.00 |
| Fairbanks | 135.00-163.00 | 191.00 | 220.00 | do | do | do | 150.00 | 202.00 | 225.00 | 135.00 | 135.00 | 150.00-185.00 | 150.00 | 115.00 |
| Koyukuk mouth | 99.00-113.00 | 126.00 | 149.00 | do | do | do | 114.00 | 148.00 | 171.00 | 99.00 | 99.00 | 114.00-131.00 | 124.00 | 98.00 |
| Nulato | 100.00-115.00 | 128.00 | 150.00 | do | do | do | 115.00 | 150.00 | 173.00 | 100.00 | 100.00 | 115.00-133.00 | 125.00 | 99.00 |
| Anvik | 104.00-120.00 | 136.00 | 158.00 | do | do | do | 119.00 | 156.00 | 179.00 | 104.00 | 104.00 | 119.00-139.00 | 134.00 | 106.00 |

[a] See special list following this table.

[b] The winter passenger rate from Seattle to Dawson per steamer, rail, and stage, is $150. From White Horse, Yukon Territory, is $100.

Beyond Dawson there is no regular winter transportation.

Winter rates for sledding on the Yukon are approximately:

| | Per ton. |
|---|---|
| Dawson to Eagle | $100.00 |
| Dawson to Circle | 240.00 |
| Eagle to Circle | 140.00 |

The following rates from Seattle to Dawson, via the White Pass route, on commodities especially applicable to placer mining, are also given for summer traffic between July 1 and August 15:

*Freight rates from Seattle to Dawson, via White Pass route, from July 1 to August 15.*

| | | |
|---|---|---|
| Lumber, rough (including flume lumber or lumber dressed on one side only) | per M feet | $48.75 |
| Oats and feed, in straight or mixed carloads | per ton | 55.00 |
| Hay (compressed to at least 22 pounds per cubic foot) 7 tons or over | do | 52.50 |
| Blacksmith's coal, 20,000 pounds and over | do | 47.50 |
| Coal oil, 20,000 pounds and over | do | 47.50 |
| Candles, 20,000 pounds and over | do | 55.00 |
| Nails, horseshoes, bolts, nuts, and spikes, 20,000 pounds and over | do | 47.50 |
| Flour, 20,000 pounds and over | do | 52.00 |
| Beef in barrels | do | 62.50 |
| Iron and iron articles: Angle, band, bar, boiler, rod, hoop, sheet and galvanized iron, sheet steel, T rails, wire rope, bolts, anvils, nuts, nails, chains, per ton | | 52.00 |
| Hardware supplies: Asbestos cement, asphaltum, axle grease, bags and bagging, cement, clay (fire), handles, oil (lubricating in cases), pulleys, iron pumps (hand), rope and cordage, tar, tin plate, varnish, wheelbarrows, per ton | | 55.00 |

TABLE 19.—*Through freight and passenger tariff for 1904, between San Francisco, Seattle, and Yukon, Koyukuk, and Tanana River points, via St. Michael.*

| | North bound. | | | South bound. | | |
|---|---|---|---|---|---|---|
| | Passengers. | | Freight per ton of 2,000 pounds, or 40 cubic feet, ship's option. | Passengers. | | Freight per ton of 2,000 pounds, or 40 cubic feet, ship's option. |
| | First class. | Second class. | | First class. | Second class. | |
| Anvik | $103.00 | $70.00 | $40.00 | $97.00 | $66.00 | $35.00 |
| Nulato | 112.50 | 80.00 | 50.00 | 105.00 | 70.00 | 40.00 |
| Koyukuk mouth | 112.50 | 80.00 | 50.00 | 105.00 | 70.00 | 40.00 |
| Fort Gibbon | 125.00 | 88.00 | 55.00 | 112.50 | 75.00 | 45.00 |
| Fairbanks | 150.00 | 110.00 | 100.00 | 125.00 | 90.00 | 75.00 |
| Rampart | 127.50 | 90.00 | 57.00 | 115.00 | 77.50 | 48.00 |
| Fort Yukon | 135.00 | 95.00 | 62.00 | 120.00 | 85.00 | 53.00 |
| Circle | 135.00 | 95.00 | 65.00 | 120.00 | 85.00 | 55.00 |
| Eagle | 140.00 | 100.00 | 70.00 | 125.00 | 90.00 | 58.00 |
| Fortymile | 140.00 | 100.00 | 70.00 | 125.00 | 90.00 | 59.00 |
| Dawson | 140.00 | 100.00 | 70.00 | 125.00 | 90.00 | 60.00 |
| Bergman | 150.00 | ........ | 100.00 | 130.00 | ........ | 75.00 |
| Bettles | 165.00 | ........ | 135.00 | 140.00 | ........ | 95.00 |

*Heavy pieces.*—On shipments to and from all river points, single pieces or packages weighing between 750 and 1,000 pounds, add 20 per cent; between 1,000 and 1,500 pounds, add 30 per cent; between 1,500 and 2,000 pounds, add 40 per cent; between 2,000 and 3,000 pounds, add 60 per cent; between 3,000 and 4,000 pounds, add 80 per cent; between 4,000 and 5,000 pounds, double rate; over 5,000 pounds taken only by special contract.

## LOCAL FREIGHT AND PASSENGER TARIFFS.

TABLE 20.—*Freight and passenger rates, for 1904, via river steamer from Dawson to points on Yukon and Tanana rivers. Return rates from 10 to 30 per cent higher.*

| Dawson to— | Freight per ton of 40 cubic feet or 2,000 pounds. | Passengers. | |
|---|---|---|---|
| | | First class. | Second class. |
| Fortymile, 53 miles | $10.00 | $7.00 | $5.00 |
| Eagle, 102 miles | 12.00 | 9.00 | 7.00 |
| Circle, 292 miles | 18.00 | 17.00 | 13.00 |
| Fort Yukon, 377 miles | 21.00 | 21.00 | 16.00 |
| Rampart, 620 miles | 26.00 | 31.00 | 23.00 |
| Fairbanks, 1,000 miles | 70.00 | 70.00 | 50.00 |
| Fort Gibbon, 700 miles | 28.00 | 34.00 | 25.00 |
| Nulato, 991 miles | 35.00 | 45.00 | 34.00 |
| Anvik, 1,196 miles | 39.00 | 54.00 | 41.00 |
| St. Michael, 1,601 miles | 48.00 | 70.00 | 50.00 |

Lumber, one and one-half the above rates per 1,000 feet B. M.

Horses and cattle, 1 to 5, one and one-half the above rates for each head; 6 to 10, one and one-quarter rate each head; over 10, one rate each head.

See footnote table 19 in regard to shipment of heavy pieces.

TABLE 21.—*Local passenger and freight tariff on Tanana River.*

PASSENGER TARIFF.

| Upstream. | | | | | | Downstream. | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| From—<br>To— | Baker Creek. | Tolovana. | Neenana. | Chena. | Fairbanks. | From—<br>To— | Chena. | Neenana. | Tolovana. | Baker Creek. | Tanana. |
| Miles from Tanana | 80 | 115 | 180 | 285 | 300 | Miles from Fairbanks | 15 | 120 | 185 | 220 | 300 |
| Tanana | $20 | $25 | $30 | $40 | $40 | Fairbanks | $5 | $15 | $20 | $25 | $25 |
| Baker Creek | .... | 10 | 20 | 30 | 35 | Chena | .... | 10 | 15 | 20 | 25 |
| Tolovana | .... | .... | 13 | 20 | 30 | Neenana | .... | | 10 | 15 | 20 |
| Neenana | .... | .... | ... | 20 | 25 | Tolovana | .... | | .... | 5 | 15 |
| Chena | .... | .... | ... | .... | 10 | Baker Creek | .... | .... | .... | | 10 |

**TABLE 21.**—*Local passenger and freight tariff on Tanana River*—Continued.

FREIGHT TARIFF.[a]

| Upstream. | | | | | | Downstream. | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| To— / From— | Baker Creek. | Tolovana. | Neenana. | Chena. | Fairbanks. | To— / From— | Chena. | Neenana. | Tolovana. | Baker Creek. | Tanana. |
| Miles from Tanana... | 80 | 115 | 180 | 285 | 300 | Miles from Fairbanks | 15 | 120 | 185 | 220 | 300 |
| Tanana | $20 | $25 | $35 | $45 | $45 | Fairbanks | $15 | $20 | $25 | $30 | $30 |
| Baker Creek | .... | 15 | 25 | 35 | 40 | Chena | .... | 15 | 20 | 25 | 30 |
| Tolovana | .... | .... | 20 | 30 | 35 | Neenana | .... | .... | 15 | 20 | 25 |
| Neenana | .... | .... | .... | 25 | 30 | Tolovana | .... | .... | .... | 15 | 20 |
| Chena | .... | .... | .... | .... | 20 | Baker Creek | .... | .... | .... | .... | 15 |

[a] Per ton of 2,000 pounds, or 40 cubic feet, ship's option.

**TABLE 22.**—*Local passenger tariff on Koyukuk River, season 1904.*

UPSTREAM.

| To— / From— | Koyukuk mouth. | Dagitli River. | Dulbi River. | Seattle Point. | Hog River. | Rock Island Point. | Red Mountain. | Bergman. | Allen River. | Peavy. | Bettles. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Miles from Nulato | 20 | 152 | 175 | 240 | 315 | 390 | 424 | 460 | 470 | 508 | 540 |
| Nulato | $5 | $16 | $18 | $24 | $34 | $44 | $48 | $51 | $54 | $60 | $66 |
| Koyukuk mouth | .... | 13 | 16 | 22 | 32 | 42 | 46 | 50 | 53 | 59 | 65 |
| Dagitli River | .... | .... | 5 | 11 | 21 | 31 | 35 | 40 | 43 | 49 | 54 |
| Dulbi River | .... | .... | .... | 8 | 18 | 28 | 32 | 37 | 40 | 46 | 52 |
| Seattle Point | .... | .... | .... | .... | 10 | 21 | 25 | 30 | 33 | 39 | 45 |
| Hog River | .... | .... | .... | .... | .... | 10 | 16 | 21 | 24 | 30 | 37 |
| Rock Island Point | .... | .... | .... | .... | .... | .... | 8 | 13 | 16 | 21 | 28 |
| Red Mountain | .... | .... | .... | .... | .... | .... | .... | 8 | 13 | 17 | 24 |
| Bergman | .... | .... | .... | .... | .... | .... | .... | .... | 5 | 15 | 20 |
| Allen River | .... | .... | .... | .... | .... | .... | .... | .... | .... | 10 | 15 |
| Peavy | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | 10 |

TABLE 22.—*Local passenger tariff on Koyukuk River, season 1904*—Continued.

DOWNSTREAM.

| To— \ From— | Peavy. | Allen River. | Bergman. | Red Mountain. | Rock Island Point. | Hog River. | Seattle Point. | Dulbi River. | Dagitli River. | Koyukuk mouth. | Nulato. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Miles from Bettles | 32 | 70 | 80 | 116 | 150 | 225 | 300 | 365 | 388 | 520 | 540 |
| Bettles | $5 | $9 | $10 | $15 | $20 | $26 | $34 | $38 | $40 | $50 | $51 |
| Peavy | .... | 5 | 6 | 12 | 17 | 22 | 30 | 34 | 36 | 47 | 48 |
| Allen River | .... | .... | 3 | 8 | 13 | 18 | 26 | 30 | 32 | 43 | 44 |
| Bergman | .... | .... | .... | 5 | 9 | 14 | 22 | 26 | 29 | 40 | 41 |
| Red Mountain | .... | .... | .... | .... | 5 | 10 | 18 | 22 | 24 | 34 | 35 |
| Rock Island Point | .... | .... | .... | .... | .... | 8 | 14 | 18 | 20 | 30 | 31 |
| Hog River | .... | .... | .... | .... | .... | .... | 8 | 12 | 15 | 24 | 25 |
| Seattle Point | .... | .... | .... | .... | .... | .... | .... | 6 | 7 | 17 | 18 |
| Dulbi River | .... | .... | .... | .... | .... | .... | .... | .... | 4 | 14 | 15 |
| Dagitli River | .... | .... | .... | .... | .... | .... | .... | .... | .... | 11 | 12 |
| Koyukuk mouth | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | 5 |

TABLE 23.—*Packing rates per 100 pounds from Rampart, Alaska, to various points in 1904.*

| | Distance. | Winter.[a] | Summer. |
|---|---|---|---|
| | *Miles.* | | |
| Hunter Creek | 4 | $2.00 | $4.00 |
| Little Minook Creek | 6 | 2.00 | 4.00 |
| Hoosier Creek | 8 | 2.00 | 4.00 |
| Ruby Creek | 10 | 2.00 | 6.00 |
| Slate Creek | 13 | 2.00 | 8.00 |
| Over divide to Pioneer Glen, Rhode Island, Omega, and Thanksgiving creeks | 25 to 32 | 6.00 | 15.00 |

[a] It should be said in regard to winter transportation that horses are gradually replacing dogs in all parts of Alaska. The dog is theoretically cheaper to feed than the horse. He gives more trouble, is more likely to give out on the trail, and experience has shown that in regions where horse feed can be obtained, and where much sledding is done, horses are cheaper.

*Circle-Fairbanks packing rates.*—The town of Circle, being the nearest point on Yukon River to the placer districts of Birch Creek and Fairbanks, has some importance, especially in winter, as a distributing center for those districts. The rates for packing in summer by horses and in winter by dog sleds from Circle are given in table 24.

TABLE 24.—*Packing rates per 100 pounds from Circle to various points.*

| | Distance. | Winter. | Summer. |
|---|---|---|---|
| | *Miles.* | | |
| Deadwood Creek | 60 | $3.00 | $25.00 |
| Eagle Creek | 80 | 6.00 | 30.00 |
| Fairbanks Creek | 135 | 15.00 | 75.00 |
| Fairbanks | 150 | 35.00 | 100.00 |
| Fort Gibbon to Cleary Creek | ........ | 60.00 | .......... |

*Packing rates at Fairbanks.*—These summer rates are continually being reduced as the trails are improved, but the following may be regarded as those that will obtain in 1905:

TABLE 25.—*Packing rates per 100 pounds from Fairbanks to various points.*

| | Distance. | Winter. | Summer. |
|---|---|---|---|
| | *Miles.* | | |
| Pedro Creek | 15 | $2.00 | $10.00 |
| Chatham Creek | 20 | 2.50 | 10.00 |
| Cleary Creek | 22 | 2.50 | 15.00 |
| Upper Fairbanks Creek | 23 | 3.00 | 20.00 |
| Lower Fairbanks Creek | 25 | 3.00 | 25.00 |

In the summer of 1904 six-horse teams were engaged in hauling loads not exceeding 1½ tons from Fairbanks to the creeks. The conditions under which this hauling was done may be seen from Pl. XL.

The nearest American supply point to the creeks of the Fortymile region is Eagle, but this town, on account of its remoteness and the absence of good roads, is at present of only subordinate importance to the miners. Dawson, although in Canadian territory, is a more convenient place to procure supplies. These are bought generally in the fall and freighted to the creeks on sleds with dogs or horses during the winter, either by way of Sixtymile Creek or Fortymile Creek. In the summer freight is carried by pack train from Eagle or by boats from the Yukon up Fortymile Creek. The winter rates from Eagle in 1903 averaged about $5 per 100 pounds; the summer rates about $25.

Freighting to the new placer district of the Porcupine is done by wagon haul across Chilkat River to Portage Point, thence to Klukwan in canoes, which are poled upstream, sometimes under great difficulties. From Klukwan the supplies are hauled up the Klehini, 15 miles, to the town of Porcupine. The rate from Haines Mission to Porcupine is $3.50 per 100 pounds, and to Pleasant Camp, 6 miles beyond Porcupine, it is $5 per 100 pounds.

Freight for the Chistochina district must be packed on horses from Valdez, via the military road to the Chistochina, and then up the river. The rate in 1903 was, in summer, $10 per 100 pounds, and in winter $3.50 per 100 pounds.

A railway has been projected from Resurrection Bay on the Alaska coast up the Sushitna and down the valley of Cantwell River to the Tanana, thence across to the Yukon at Rampart. A few miles of the road have actually been constructed from the Resurrection Bay terminus. There is little doubt that a railway following either this route or one following the already surveyed route from Valdez to Tanana Crossing would pay as an investment. The benefit of such lines of railway to the south interior portion of Alaska, including the Hope and Sunrise, Chistochina, and Fairbanks placer districts, is unquestionable.

Freight rates from Hope and Sunrise to the mining creeks in the vicinity are as follows:

Supplies are freighted from Sunrise to Mills Creek, about 18 miles, for 5 cents per pound.

From Sunrise to Lynx Creek, between 20 and 25 miles, for about the same. This is contract work.

Supplies are boated from Sunrise to mouth of Glacier Creek (7 to 8 miles) and there packed to Crow Creek (6 to 7 miles) for 3 to 3½ cents, but this is by private yearly contract. The miners usually bring their own supplies over the snow in the winter time so that there is not much freighting.

The transportation of supplies from supply points on Seward Peninsula to the mining creeks is done mainly by hauling in wagons; in some cases, as from Cheenik to Council, by flatboat, and in three cases, Nome-Anvil Creek, Solomon-East Fork, and Council Number 15, Ophir Creek, by railway. Between points on the coast the summer transportation depends on small ocean-going boats, for the most part gasoline launches, whose trips are not regular, as landings at Nome, Solomon, Bluff, and York depend on the weather. All supplies larger than those which can be transported in dories must be lightered at the above-named points. Charges for lightering at Nome are included in table 26. It should be understood that the ocean rates to Seward Peninsula points given in table 17 include the charge for lightering in every case.

The wagon haul is fairly easy at some times of the year in the broad, gravel-filled river beds. These streams are subject to flood, however, and such teaming is impossible at times. From White Mountain, on Fish River, to Council, Ophir Creek, Gold Bottom, and as far as Goose Creek, on the Casadepaga, a useful type of flatboat, drawn by a strong horse, draws 5 tons during favorable conditions of water. Such hauling, when possible, reduces the freight over the wagon haul by about 40 per cent. Such horse boats can ascend the Casadepaga only during June and early July.

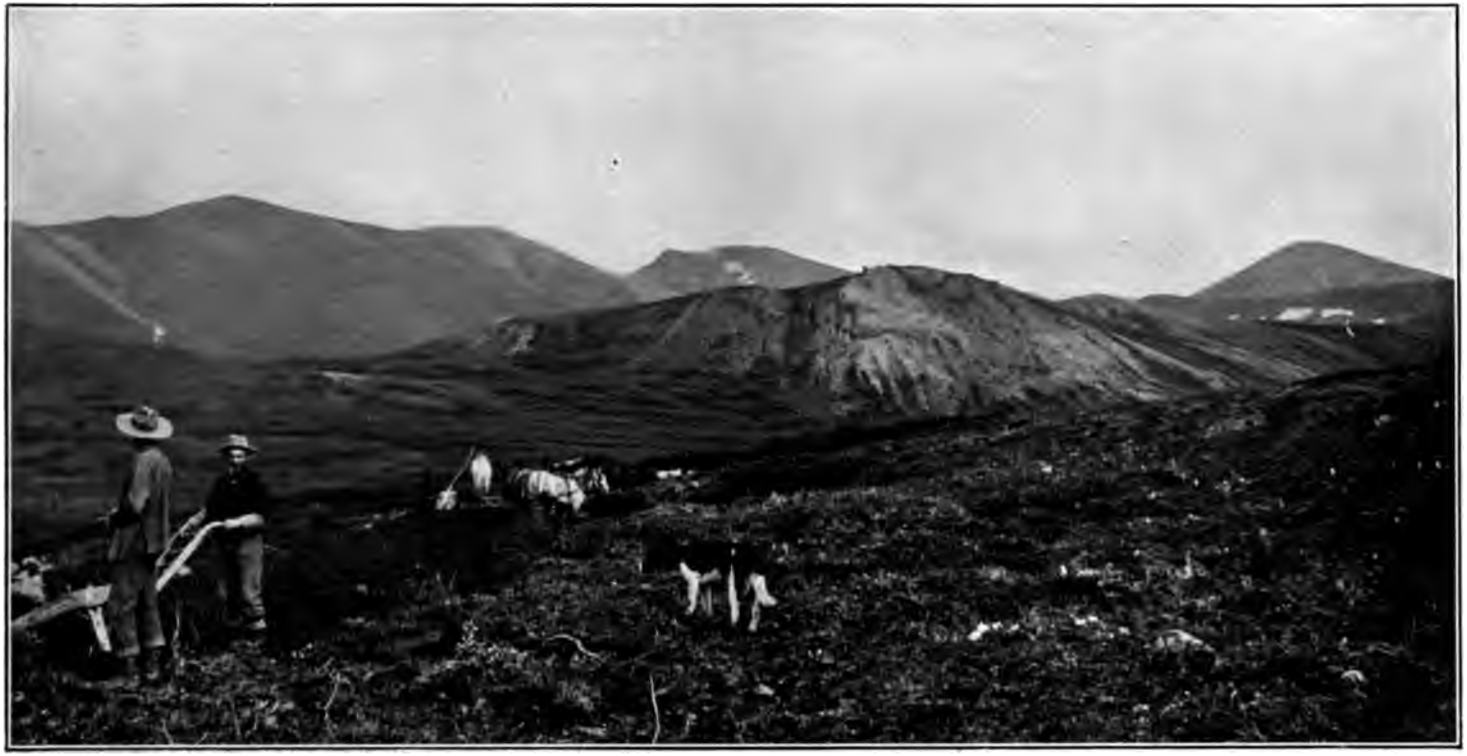
The freight rates from Teller to the creeks of the Kougarok district were not available in detail for this report. Freighting by wagon haul is, however, carried on from Teller, York, and some of the other more distant coast points of Seward Peninsula, and the rates may be figured as higher than at Nome in proportion to the increased cost of supplies at those coast points.

On Ophir Creek, in the Council district, it has been shown that the larger companies can reckon on obtaining their supplies direct from San Francisco via ocean steamer to Cheenik; flat steamers, Cheenik to mouth of the Niukluk; horse boat, Niukluk mouth to Council, and railway from Council to the creek, at a cost of approximately 2 cents a pound in excess of San Francisco prices. Table 26 gives the local freight rates from the main supply points to the creeks.

TABLE 26.—*Summer local freight and passenger rates from Nome to various points in Seward Peninsula, 1904.*

| Point. | Method of transportation. | General freight. | Lumber. | Horses and cattle. | Passengers. |
|---|---|---|---|---|---|
| | | *Per ton.* | *Per M ft.* | *Per head.* | |
| Solomon[a] | Steamer | $15.00 | $22.50 | $18.00 | $7.50 |
| Bluff[a] | do | 15.00 | 22.50 | 18.00 | 10.00 |
| Cheenik | do | 15.00 | 22.50 | 22.50 | 15.00 |
| Council | Steamer and flatboat. | 27.50 | 35.00 | 35.00 | 20.00 |
| Teller[b] | Steamer | 15.00 | 22.50 | 22.50 | 15.00 |
| York[a] | do | 20.00 | 30.00 | 30.00 | 20.00 |
| Kiwalik | do | 30.00 | 15.00 | 45.00 | 30.00 |
| Deering[a] | do | 30.00 | 45.00 | 45.00 | 30.00 |
| Anvil Creek | Wagon haul, railway.[c] | $10.00–15.00 | | | |
| Cooper Gulch | Wagon haul | 10.00 | | | |
| Moonlight Gulch | do | 10.00 | | | |
| Bourbon Creek | do | 10.00 | | | |
| Dry Creek | do | 5.00–10.00 | | | |
| Holyoke Creek | do | 10.00–15.00 | | | |
| Dexter Creek | do | 2.00 | | | |
| Buster Creek | do | 30.00 | | | |
| Lillian Creek | do | 30.00 | | | |
| Osborne Creek | do | 30.00–40.00 | | | |
| Seattle Creek | do | 37.50 | | | |
| Washington Creek | do | 30.00 | | | |
| Moss Gulch Creek | do | 30.00 | | | |

[a] No harbor. [b] Nome to Lanes landing via Kuzitrin River, $50 per ton. [c] Wagon haul, 1½-ton rates.



TOPOGRAPHY, SEWARD PENINSULA PROVINCE.

TABLE 26.—*Summer local freight and passenger rates from Nome to various points in Seward Peninsula, 1904*—Continued.

| Point. | Method of transportation. | General freight. | Lumber. | Horses and cattle. | Passengers. |
|---|---|---|---|---|---|
| | | *Per ton.* | *Per M ft.* | *Per head.* | |
| Peluk Creek | Wagon haul | $5.00–10.00 | | | |
| Otter Creek | do | 10.00–15.00 | | | |
| Martin Creek | do | 10.00–15.00 | | | |
| Glacier Creek | do | 30.00 | | | |
| Rock Creek | do | 30.00 | | | |
| Prospect Creek | do | 40.00 | | | |
| Lindbloom Creek | do | 40.00 | | | |
| Alpine Creek | do | 40.00 | | | |
| Sledge Creek | do | 40.00 | | | |
| Bolto Creek | do | 50.00 | | | |
| Bangor Creek | do | 60.00 | | | |
| Twin Mountain Creek | do | 60.00 | | | |
| Pioneer Creek | do | 60.00 | | | |
| Last Chance | do | 60.00 | | | |
| Penny River | do | 20.00–30.00 | | | |
| Willow Creek | do | 30.00 | | | |
| Oregon Creek | do | 100.00 | | | |
| Nugget Creek | do | 100.00 | | | |
| Cripple River | do | 40.00–50.00 | | | |
| Nome River, head | do | 100.00 | | | |
| Fountain Creek | do | 30.00 | | | |
| East Fork Solomon to Council. | Stage line | | | | $12.50 |

*Rates for lightering cargo at Nome.*

| | | |
|---|---|---|
| General merchandise, including small machinery | per ton | $5.00 |
| Machinery: | | |
| Two- to three-ton pieces | do | 7.50 |
| Three- to four-ton pieces | do | 10.00 |
| Four- to five-ton pieces | do | 15.00 |
| Five- to six-ton pieces | do | 20.00 |
| Six- to seven-ton pieces | do | 25.00 |
| Seven- to eight-ton pieces | do | 30.00 |
| Double-compressed hay | do | 5.00 |
| Coal in sacks | do | 3.50 |
| Coal put on beach and piled up | do | 4.00 |
| Lumber | per M feet | 5.50 |
| Live stock | per head | 6.00 |

Winter freighting, by means of horse and dog sleds, may be reckoned at about one-half the summer hauling rates.

From Nome to Council, 85 miles, winter haul is—

| | |
|---|---|
| Warm-storage rate, per ton | $60.00 |
| Cold-storage rate, per ton | 40.00 |
| Passenger rate, forty hours | 15.00 |

*Summer local freight rates from Solomon to various points.*

| | Per ton. |
|---|---|
| Mouth Shovel Creek | $8.00-$12.00 |
| Hurrah Creek[a] | 10.00 |
| Willow Creek | 90.00 |
| Ruby Creek | 80.00 |
| Mystery Creek | 15.00 |
| West Creek | 12.50 |
| East Fork Creek | 15.00 |
| Down coast to Spruce Creek | 12.50 |

*Summer local freight rates from Council to various points.*

| | Per ton. |
|---|---|
| No. 15, Ophir Creek (railway) | $10.00 |
| Gold Bottom (horse boat) | 30.00 |
| Gold Bottom (wagon haul) | 50.00 |
| Lower Casadepaga | 75.00 |

## CUSTOMS REGULATIONS.

Considerable delay and confusion have been caused in the past by shippers not understanding the requirements of the United States and Dominion customs departments. The following information is given for the guidance of shippers, and the instructions herein contained must be strictly adhered to:

Shipments of goods from United States ports via Skagway, destined to Eagle City and other lower Yukon River points in transit through British Columbia and Yukon Territory, must be accompanied by 4 certified copies of invoices and 2 copies of bills of lading. All of these papers are necessary for customs purposes at Skagway.

Goods shipped from United States ports, destined to points in Alaska via Skagway and Yukon River, are bonded through Canadian territory duty free.

Charges for preparing customs papers by the customs agent of the White Pass and Yukon route will be as follows: For preparing shipper's manifest, 50 cents; transportation and exportation entries will be charged for on basis of actual cost to the White Pass and Yukon route.

The customs agent will notify the agent of the White Pass and Yukon route at Skagway station of the amounts to be assessed on above account, and the same will be placed against each shipment as a back charge.

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[a] Under exceptionally favorable conditions large lots have been landed San Francisco-Little Hurrah Creek as low as $17 per ton.

Difficulties of a peculiar nature are experienced by the placer miners of interior Alaska owing to the configuration of the international boundary between the Territory of Alaska and the British possessions. The town of Dawson, being the largest settlement of the interior, is naturally the point where mining supplies are most easily and quickly obtained by the placer miners of the neighboring American camps of Birch Creek and Fortymile, the Rampart district, and others of the Yukon and Tanana. During the season of 1904 the rapid and to a certain extent justifiable development of the town of Fairbanks, on Tanana River, gave promise that this settlement would eventually become as well equipped to supply the needs of the Tanana miners as is Dawson. The American Yukon camps must, however, for a considerable number of years be locally dependent on Dawson as a source of supplies.

Theoretically the miners are supposed to buy their mining supplies and machinery either at points in the United States or at towns on the Yukon in American territory. Such supplies, of course, if shipped down the Yukon, come through the Canadian territory in bond free of duty. In practice, however, owing to the necessity of obtaining supplies quickly, purchases are made in Dawson, and unless the supplies can be proved to be returned American goods the American miner must pay duties to the United States custom-house at Eagle according to the rates given in the following list. Frequently duties must be paid even on supplies made in the United States and returned, since there is no manufacturer's mark by which the customs inspector can identify them. This especially applies to such important mining supplies as small parts of boilers, tubes, pipe, pipe fittings, elbows, T's, unions, grate bars, sledges, drill steel, picks, shovels, pump plungers, buckets, valves, small car wheels, strap iron, rails, bar and rod iron, and small castings. Even larger pieces, such as parts of hydraulic giants, are frequently unstamped. These supplies, kept for sale by the machinery firms of Dawson, frequently pay customs duties both to Canada and the United States before reaching the Alaska camps. Dawson is frequently sought by the American miners in the midst of the short season for the purpose of repairs to various mining machinery, it being the only place accessible on the Yukon where machine shops exist. It may easily be seen how burdensome on the miner is the imposing of a duty on his machinery which he brings back into American territory after repairs, and on which he must pay a duty because it has been improved or changed in foreign territory.

Below are given the United States customs duties on small mining supplies:

*United States customs duties.*

| Articles. | Duty. |
|---|---|
| Boilers and engines, first and second hand | 45 per cent ad valorem. |
| Points (steam), sluice forks, picks, shovels, axes | 45 per cent ad valorem. |
| Sledges | 1½ cents per pound. |
| Drill steel | 4.7 cents per pound. |
| Hydraulic hose | 20 cents per pound. |
| Cast pipe | 4.7 cents per pound. |
| Hydraulic pipe | 2 cents per pound. |
| Gas pipe | 35 per cent ad valorem. |
| Rubber hose | 30 per cent ad valorem. |
| Coke | 25 per cent ad valorem. |
| Bituminous coal | 67 cents per ton. |
| Dynamite | 6 cents per pound. |
| Black powder | 6 cents per pound. |
| Caps | $2.36 per thousand. |
| Fuse | 45 per cent ad valorem. |
| Candles | 25 per cent ad valorem. |
| Nails, wire | One-half cent per pound. |
| Nails, wrought iron | 2¼ cents per pound. |
| Steel rails | Seven-twentieths cent per pound. |
| Wire cable | 1 cent per pound and 40 per cent ad valorem. |
| Centrifugal pumps | 45 per cent ad valorem. |
| Sawed lumber | $2 per thousand and 50 cents for each side dressed. |
| Wheelbarrows, iron | 45 per cent ad valorem. |
| Rope | 1 cent per pound. |
| Self-dumping carriers | 45 per cent ad valorem. |
| Asphalt coating | $3 per ton. |
| Hydraulic giants | 45 per cent ad valorem. |
| Lubricating oils | 25 per cent ad valorem. |
| Copper plates | 2½ cents per pound. |
| Cement | 8 cents per 100 pounds. |
| Quicksilver | 7 cents per pound. |
| Iron castings | Eight-tenths cent per pound. |
| Iron castings, ground and fitted | 45 per cent ad valorem. |

It should be stated that there is a duty on all horses, cattle, sheep, and dogs[a] entering Alaska. Thus a dog team coming down river from Dawson in the winter is dutiable.

During the season of 1904 a United States customs inspector accompanied the Yukon steamers from Dawson with shipments from that point to Fairbanks, in order to obviate the long delays which had previously been necessary for appraisement at Eagle.

Owing to the export duty on gold which is charged by the Canadian government, some difficulties have been experienced by Americans who bring gold mined in Alaska Territory through Canadian territory, via Dawson, en route to the United States. The Canadian government naturally assumes that all gold found on south-bound passengers leaving Canadian territory at White Pass has been mined in Canada. Such gold is therefore liable to confiscation by the officers in charge of collection of royalty unless the bearer is provided with a properly ratified certificate stating that the gold is of foreign production. American miners should obtain such a certificate at Eagle from the United States authorities (fee, 10 cents) and have the same ratified at Dawson by the Canadian police (fee, 50 cents for gold from 1 to 25 ounces, $1 for 25 ounces and upward). All individuals leaving Yukon Territory for United States territory are strictly searched for gold.

The regulations in regard to boats coming down the Yukon into United States territory must also be taken into consideration. No boat or scow, not a regular carrier, can, for example, make the journey from Dawson to Circle, carrying over 5 tons of freight. This is especially burdensome on settlers who wish to outfit at Dawson and float their goods down the Yukon. The situation, owing to the conditions, is undoubtedly a burden of extra expense on the miner, but is unavoidable, and must be reckoned with in any estimate of expenses attending the prosecution of placer mining in the Alaska interior.

## TELEGRAPH RATES.

By means of the United States military telegraph line in Alaska, including the wireless system from St. Michael to Port Safety and the Seattle-Sitka-Valdez cable with connection to Juneau and Skagway, an all-American service is now maintained from all important points of Alaska (except Circle) to the outside world.

The rates for the military service in Alaska are based upon a tariff of 2 cents per word for each 100 miles or fraction thereof, address and signature not counted, a message of less than 10 words being charged for as a 10-word message. During the month of September, 1904, a cable line was completed between Sitka and Valdez, which, with the Sitka-Seattle cable, completes the American telegraph service to all parts of Alaska. Through 10-word rates from Seattle are as follows:

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[a] Horses, value $150 or less, $30 per head; over $150, 25 per cent ad valorem. Dogs, 20 per cent ad valorem; cattle, $2 per head, under 1 year old; others, 27½ per cent ad valorem.

*Telegraph rates from Seattle to points in Alaska.*

| | |
|---|---|
| Sitka | $2.50 |
| Juneau | 2.75 |
| Skagway | 3.00 |
| Valdez | 3.50 |
| Nome | 4.00 |
| Eagle | [a]3.00 |
| Rampart | [a]3.50 |
| Fairbanks | [a]3.50 |

## DISTANCES AND COST OF SUPPLIES.

Tables Nos. 27, 28, and 29 give approximate distances in Alaska.

TABLE 27.—*Distances, in miles, between towns in Alaska, excluding Seward Peninsula.*

| | Juneau. | Valdez. | Haines. | St. Michael. | Nulato. | Fort Yukon. | Rampart. | Fort Gibbon. | Fairbanks. | Circle. | Eagle. | Fortymile. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Valdez | 580 | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Haines | 100 | 600 | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| St. Michael | 1,880 | 1,300 | 1,900 | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Nulato | 1,140 | 1,720 | 1,249 | 485 | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Fort Yukon | 963 | 1,543 | 864 | 990 | 505 | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Rampart | 203 | 1,783 | 984 | 750 | 265 | 240 | ...... | ...... | ...... | ...... | ...... | ...... |
| Fort Gibbon | 1,268 | 1,848 | 1,049 | 685 | 200 | 305 | 65 | ...... | ...... | ...... | ...... | ...... |
| Fairbanks | 1,423 | 2,003 | 1,204<br>[b]920 | 840 | 355 | 455 | 220 | 155 | ...... | ...... | ...... | ...... |
| Circle | 878 | 1,458 | 779 | 1,075 | 590 | 85 | 325 | 270 | [c]150 | ...... | ...... | ...... |
| Eagle | 745 | 1,325 | 646 | 1,208 | 723 | 218 | 458 | 523 | [c]283 | 135 | ...... | ...... |
| Fortymile | 695 | 1,275 | 596 | 1,253 | 768 | 263 | 503 | 568 | 328 | 180 | 45 | ...... |
| Dawson | 647 | 1,227 | 548 | 1,298 | 813 | 308 | 548 | 613 | 373 | 225 | 90 | 45 |

[a] 35 cents each additional word.

[b] Taking trail at Circle.

[c] By trail.

TABLE 28.—*Distances, in miles, between points on Seward Peninsula.*

| | Nome. | Cheenik. | Council. | Bluff. | Solomon. | Teller. | York. | Kewalik. | Blossom. |
|---|---|---|---|---|---|---|---|---|---|
| Cheenik | 90 | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Council | 135 | 45 | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Bluff | 65 | 25 | 70 | ...... | ...... | ...... | ...... | ...... | ...... |
| Solomon | 34 | 56 | 101 | 31 | ...... | ...... | ...... | ...... | ...... |
| Teller | 100 | 190 | 235 | 165 | 134 | ...... | ...... | ...... | ...... |
| York | 110 | 200 | 245 | 175 | 144 | 45 | ...... | ...... | ...... |
| Kiwalik | 345 | 435 | 480 | 410 | 379 | 280 | 235 | ...... | ...... |
| Blossom | 310 | 400 | 445 | 375 | 344 | 245 | 200 | 55 | ...... |
| Candle | 360 | 450 | 495 | 425 | 394 | 295 | 250 | 7 | 62 |

TABLE 29.—*Distances from main supply points in Alaska to neighboring creeks.*

FROM CIRCLE.

| | Miles. |
|---|---|
| Diggings on Deadwood Creek | 40 |
| Mouth of Boulder Creek | 35 |
| Junction of Mastodon and Independence creeks | 47 |
| Mouth of Miller Creek | 45 |
| Junction of Mastodon Fork and Eagle Creek | 53½ |

FROM FAIRBANKS.

| | |
|---|---|
| Gilmore at crossing of trail | 14 |
| Junction of Twin and Pedro creeks | 18 |
| Junction of Wolf and Cleary creeks | 24 |
| Fairbanks Creek at mouth of Alder Gulch | 27 |

FROM NOME.

| | |
|---|---|
| Discovery on Anvil Creek | 4 |
| No. 3 above on Glacier Creek | 7½ |
| Junction of Grass Gulch and Dexter Creek | 7 |
| Cooper Gulch | 4 |
| Bourbon Creek (passes through Nome). | |
| Holyoke Creek | 2½ |
| Dexter Creek | 7 |
| Buster Creek | 9 |
| Lillian Creek | 11 |
| Osborne Creek | 7 |
| Seattle Creek | 22 |
| Washington Creek | 6 |
| Moss Gulch | 6½ |
| Peluk Creek | 2 |
| Otter Creek | 3½ |
| Martin Creek | 2½ |
| Rock Creek | 3¾ |
| Prospect Creek | 12 |
| Lindebloom Creek | 11 |
| Alpha Creek | 11½ |

| | Miles. |
|---|---|
| Sledge Creek | 13 |
| Balto Creek | 12½ |
| Bangor Creek | 17 |
| Twin Mountain Creek | 15 |
| Last Chance Creek | 20 |
| Penny River | 11 |
| Willow Creek | 10 |
| Oregon Creek | 19 |
| Nugget Creek | 18 |
| **FROM SOLOMON.** | |
| Mouth of Shovel Creek | 5 |
| Mouth of Kasson Creek | 10 |
| Mouth of Big Hurrah Creek | 8½ |
| Mouth of East Fork | 11 |
| Mouth of Coal Creek | 15 |
| Head of Ruby Creek (Casadepaga drainage) | 20 |
| Head of Penelope Creek (Casadepaga drainage) | 21 |
| Mouth of Goose Creek (Casadepaga drainage) | 25 |
| **FROM COUNCIL.** | |
| Mouth of Ophir Creek | 2½ |
| Mouth of Dutch Creek | 8 |
| Mouth of Crooked Creek | 11 |
| Mouth of Basin Creek | 2 |
| Junction of Gold Bottom and Warm Creek | 8 |
| Mouth of Mystery Creek | 4 |
| **FROM TELLER.** | |
| Bering Creek | 13 |
| Mouth of Right Fork of Bluestone River | 15 |
| Mouth of Alder Creek | 17 |
| Mouth of Ruby Creek | 13 |
| Mouth of Sunset Creek | 3½ |
| Marys Igloo | 50 |
| **FROM MARYS IGLOO.** | |
| Quartz Creek | 24 |
| Angeles Creek | 33 |
| Goodall Creek | 43 |
| Homestake Creek | 52 |
| Macklin Creek | 56 |
| North Fork of Kougarok River | 21 |
| Harris Creek | 20 |
| Mouth of Baldy Creek | 20 |
| Mouth of Portage Creek | 22 |

TABLE 30.—*Prices of mining supplies in the North.*

| | Juneau. | Dawson.[a] | Circle. | Mastodon post-office, Birch Creek region. | | Fairbanks. | Nome. |
|---|---|---|---|---|---|---|---|
| | | | | Summer. | Winter. | | |
| **Asphalt coating** | 8 cts. per gal. | $2 per gal. | | | | | 75 cts per gal. |
| **Boilers:** | | | | | | | |
| 50-horsepower return-flue, American manufacture. | | $1,800 | | | | $3,000 [b] | (c) |
| 40-horsepower return-flue | | $1,600 | | | | $2,500 | |
| 25-horsepower return-flue | | $1,200 | | | | $1,800 | |
| 10-horsepower return-flue | | $600 | | | | $1,100 | |
| 5-horsepower return-flue | | $300 | | | | $500 | |
| **Caps** | 98 cts. per 100 | $1.50 per 100 | | | | $1.75 per 100 | $1.50 per box. |
| **Candles** | $2.40 per box | $3.75 per box | $4 per box | $6 per box | $4.50 per box | $4 per box | $2.50 per box. |
| **Cars, self-dumping, ore** | | $45 | | | | $50 | |
| **Cement** | $4.25 per bbl | $20 per bbl | | | | $25 per bbl | $9 per bbl. |
| **Clay (fire)** | | 10 cts. per lb. | | | | 15 cts. per lb. | 15 cts. per lb. |
| **Copper plates** | $2 per sq. ft. | 50 cts. per lb. | | | | 75 cts. per lb. | $1 per sq. ft. |
| **Copper wire** | 40 cts. per lb. | 60 cts. per lb. | | | | 75 cts. per lb. | 50 cts. per lb. |
| **Coal, blacksmith** | $22 per ton | $100 per ton | $140 per ton | $25 per cwt | $10 per cwt | $120 per ton | $35 per ton.[d] |
| **Coal, fuel, local mine** | | $12 to $16 per ton | | | | | |
| **Coke** | $30 per ton | 5¼ cts. per lb. | | | | 6 cts. per lb. | |
| **Coal oil** | $2.90 per case | 50 cts. per gal. | $9 per case | $15 per case | $10 per case | 75 cts. per gal | $3.50 per case 10 gal.[e] |
| **Drilling steel** | 12 cts. per lb. | 20 cts. per lb. | 15 to 25 cts. per lb. | 25 to 35 cts | 20 to 30 cts | 20 cts. per lb. | 25 cts. per lb.[f] |

[a] Prices at Eagle are practically the same as at Dawson, plus duty on imported supplies.

[b] Includes duty in coming from Dawson, Yukon Territory; can be landed from outside to Fairbanks a little cheaper than at Dawson; portable return tubular made in Erie, Pa., or in Oil City, considered most economical for this class of work; Scotch marine type is best for dirty water used in this work, but is much heavier; 40-horsepower weighs 12,100 pounds, and about 1 ton heavier for each additional 10-horsepower; 50-horsepower weighs 11,700 pounds; 40-horsepower weighs 10,400 pounds; 25-horsepower weighs 7,300 pounds; 10-horsepower weighs 4,300 pounds; 5-horsepower weighs 1,800 pounds.

[c] Second-hand boilers, 6-horsepower, $200; 8-horsepower, $200; 10 to 15-horsepower, $250 to $300; 20-horsepower, approximately, $500; 25 to 30-horsepower, $600 to $750; 50-horsepower, $1,000.

[d] Bituminous coal, $17.25 a ton; anthracite, about $40 a ton.

[e] $4 a barrel of 42 gallons. Pearl is $3.25 a case of 10 gallons.

[f] Soft steel, 10 cents a pound; also soft iron, 10 cents a pound.

TABLE 30.—*Prices of mining supplies in the North*—Continued.

| | Juneau. | Dawson. | Circle. | Mastodon post-office, Birch Creek region. | | Fairbanks. | Nome. |
|---|---|---|---|---|---|---|---|
| | | | | Summer. | Winter. | | |
| **Engines:** | | | | | | | |
| 40-horsepower | | $1,000 | | | | $2,000 | |
| 25-horsepower | | $650 | | | | $1,500 | (a) |
| 16-horsepower | | $500 | | | | $1,000 | |
| 8-horsepower | | $300 | | | | $600 | |
| **Fittings** (steam-pipe) | | 20 cts. per lb | | | | 40 cts. per lb | (b) |
| **Fuse** | 65 cts. per 100 feet. | $1 per 100 feet | $1.50 per 100 ft | | | $2 per 100 ft | $1 per 100 ft. |
| **Gasoline** (distillate) | $3.75 per case | $9 per 10-gal | $9 to $10 per case | | | $15 per 10-gal. case | $3.50 to $4 per case. |
| **Handles** (pick and sledge) | $3 to $3.35 per doz. | $6 per doz | $7.50 to $9 per doz. | $1 each | | $12 per doz | $6 per doz. |
| **Hoists**, steam: | | | | | | | |
| 8-horsepower single-cylinder vertical. | | | | | | $550 | |
| 15-horsepower double-drum and engine. | | $1,200 | | | | $2,500 [c] | (a) |
| 12-horsepower double-drum and engine. | | $800 | | | | $2,000 | |
| **Hulls**, barge at St. Michael [d] | | | | | | | |
| **Hose**, steam: | | | | | | | |
| ½-inch 6-ply | 46 cts. per ft | 60 cts. per ft | | | | 75 cts. per ft | 60 cts. per ft. |
| ¾-inch 6-ply | 60 cts. per ft | 75 cts. per ft | | | | $1 per ft | 75 cts. per ft. |
| **Hose**, hydraulic, 12 ounces: | | | | | | | |
| 6-ply (imported) | | $1.25 per ft | | 55 cts. [e] | | $1.50 per ft | 35 cts. per ft. |
| 8-ply (imported) | | $1.50 per ft | | | | $1.75 per ft | 50 cts. per ft. |
| Hose, fire 2½-ply | | $1.50 per ft | | | | $1.50 per ft | |
| **Hydraulic** pipe: | | | | | | | |
| 6-inch [f] | | $1.25 per ft | | | | $1.50 per ft | |
| 8-inch | | $1.50 per ft | | | | $1.75 per ft | |
| K. D., 6-inch | | 90 cts. per ft | | | | $1 per ft | |
| K. D., 8-inch | | $1 per ft | | | | $1.25 per ft | |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Hydraulic giants: | | | | | | | |
| No. 0 | | $75 | | | | $100 | |
| No. 2 | | $150 | | | | $175 | |
| Iron: | | | | | | | |
| Sheet | 4 cts. per lb | 12 cts. per lb | 15 cts. per lb | 25 cts. per lb | 18 cts. per lb | | 8 cts. per lb. |
| Corrugated | | $9 per sq | 15 cts. per lb | | | $10 per sq | 7 cts. per lb. |
| Bar, galvanized | 5 cts. per lb | 10 cts. per lb | 15 cts. per lb | | | 15 cts. per lb | |
| Lumber: | | | | | | | |
| Rough | $12 per M | $50 per M | | $110 per M | | $75 per M | $32.50. per M. |
| Planed | $13 per M | $70 per M | | | | $100 per M | |
| Small stripping | | 2 cts. per lin. ft | | | | 3 cts. per lin. ft | |
| Nails, wire | $3 per keg | $8 per keg | $12.50 per cwt | 25 cts. per lb | 18 cts. per lb | $10 per keg | $6 per keg. |
| Oil, lubricating: | | | | | | | |
| Cylinder | 85 to 90 cts. per gal. | $1.75 per gal | 75 cts. per gal | | | $2 per gal | $1 per gal. |
| Machine | 35 to 40 cts. per gal. | $1.50 per gal | 50 cts. per gal | $1 per gal | | $1.50 per gal | 75 cts. per gal. |
| Petroleum, crude, $1.70 barrel; 4 barrels ton; St. Michael. | | | | | | | |
| Pans, gold | 20 cts. each | 75 cts. each | $1 each | $1.50 each | | $1 each | 50 cts. each. |
| Pipe, cast-iron: | | | | | | | |
| ½-inch | 4½ cts. per ft | 12 cts. per ft | | | | 20 cts. per ft | |
| ¾-inch | 5½ cts. per ft | 15 cts. per ft | | | | 25 cts. per ft | |
| 1-inch | 7 cts. per ft | 20 cts. per ft | | | | 30 cts. per ft | |
| Picks, without handles | $7 per doz | $22 per doz | $20 to $24 per doz | | | $25 per doz | $12 per doz. |
| Pumps: | | | | | | | |
| Force— | | | | | | | |
| 2-inch | | $175 | | | | $300 [g] | |
| 3-inch | | $350 | | | | $400 | |

[a] 25-horsepower double drum, hard to get, $1,250; 8 to 10-horsepower, $350 to $600.

[b] Small fittings same as Seattle; ¾-inch pipe, 10 cents a foot; 1-inch, 12½ cents; 1¼-inch, 15 cents; 1½-inch, 20 cents; 2-inch, 25 cents; 6-inch, $1 a foot.

[c] Made in San Francisco for self-dumpers, 50-pan steel bucket. A 6-horsepower weighs 1,000 pounds, costs $450 for a 30-pan bucket; is also obtainable at Fairbanks; 15-horsepower weighs 5,500 pounds; 12-horsepower weighs 3,200 pounds.

[d] Barge, 100 by 26 feet by 6 feet 6 inches, Oregon fir, good for 200 tons at 4 feet draft, costs between $4,000 and $5,000; cost to tow to Dawson, $1,500; to Fairbanks, $1,250.

[e] Manufactured in Dawson, 15 ounces, 9 to 14 inches diameter, for 15 to 30 cents a foot.

[f] See prices of hydraulic machinery in San Francisco, under "Hydraulic mining."

[g] 750 pounds and 1,050 pounds.

TABLE 30.—*Prices of mining supplies in the North*—Continued.

| | Juneau. | Dawson. | Circle. | Mastodon post-office, Birch Creek region. | | Fairbanks. | Nome. |
|---|---|---|---|---|---|---|---|
| | | | | Summer. | Winter. | | |
| Pumps—Continued. | | | | | | | |
| Centrifugal— | | | | | | | |
| 5-inch | | $200 | | | | $250[a] | |
| 6-inch | | $250 | | | | $300 | |
| 10-inch | | $500 | | | | $300 | |
| 8-inch | | | | | | $650 | |
| Powder: | | | | | | | |
| Dynamite | 60 per cent, 18 cts. per lb. | 35 cts. per lb. | | | | 50 cts. per lb. | |
| Black | | $2.75 per keg of 25 lbs. | 75 cts. to $1 per lb. | | | $3 per keg of 25 lbs. | |
| Quicksilver | 65 cts. per lb | $1 per lb | $1 per lb | | | $1 per lb | |
| Rockers (frame built in Dawson) | | $2.75, plates | | | | $3 | |
| Rope: | | | | | | | |
| Manila | 13 cts. per lb. | 30 cts. per lb. | $30 to $35 per cwt. | | | 40 cts. per lb. | |
| Wire— | | | | | | | |
| ⅜-inch | | 16 cts. per ft | | | | 18 cts. per ft | |
| ½-inch | | 18 cts. per ft | | | | 22 cts. per ft | |
| ⅝-inch | | 22½ cts. per ft | | | | 24 cts. per ft | |
| ¾-inch | | 30 cts. per ft | | | | 32 cts. per ft | |
| Shafting | 44 cts. per lb. | 15 cts. per lb. | | | | 20 cts. per lb. | |
| Shovels | $8.50 per doz. | $18 per doz | $20 to $24 per doz. | $3 each | $2.50 each | $20 per doz | $9 per doz. |
| Sluice forks | $1.35 each | $20 per doz | $3.50 each | $5 each | $3.50 each | $25 per doz | $18 per doz. |
| Steel plates and bars | 9 to 20 cts. per lb. | 12 cts. per lb. | 15 cts. per lb. | | | 15 cts. per lb. | 12 cts. per lb. |
| Steel rails, T, 8-pound | | 10 cts. per lb. | | | | 15 cts. per lb. | 6 cts. per lb. |
| Thawing points, one-half and three-fourth inch steel head. | | $10 each | $8 to $10 each | | | $10 each | $5.50 each. |
| Wheels for cars | | $17.50 per set | | | | $20 per set | $20 per set. |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Bolts | | | 15 cts. per lb | 20 cts. per lb | 18 cts. per lb | 25 cts. per lb | |
| Nuts | 6 cts. per lb | 20 cts. per lb | 15 cts. per lb | 20 cts. per lb | 18 cts. per lb | 25 cts. per lb | 15 cts. per lb. |
| Spikes | | 10 cts. per lb | | | | 15 cts. per lb | $6 per keg. |
| Scrapers: | | | | | | | |
| No. 1 | | $22.50 each | $10 to $20 each | $35 each | $25 each | $25 | $12. |
| No. 2 | | $20 each | | | | $22.50 | $10. |
| No. 3 | | $18 each | | | | $20 | $9. |
| Wheelbarrows, Pan-American (all steel, $15.) | $2.50 to $6.50 each. | $12.50 | $7.50 to $12.50 | $20 each | $15 each | $20 | |
| Wagons (Bain) | | $160 to $180 | | | | $200 | |
| Leather | 40 to 45 cts. per lb. | $1 per lb | $1 per lb | | | $1.25 per lb | |
| Self-dumping carriers: | | | | | | | |
| Made in San Francisco[b] | | | | | | $80 to $90 | |
| Made in Dawson | | $75 | | | | | |
| Pilot bread | $1.65 per 27-lb. box. | 15 cts. per lb | 25 cts. per lb | 40 cts. per lb | 30 cts. per lb | 20 cts. per lb | 10 cts. per lb. |
| Flour | $2.90 per 100 lbs | $6.50 per 100 lbs | $7 per cwt | $15 per cwt | $11 per cwt | $9 per 100 lbs | $3.25 per cwt. |
| Rolled oats | 45 cts. per 10-lb. sack. | 8 cts. per lb | $9 to $10 per cwt | $17 per cwt | $17 per cwt | 13½ cts. per lb | 4½ cts. per lb. |
| Corn meal | 35 cts. per 10-lb. sack. | 7 cts. per lb | $9 to $10 per cwt | $17 per cwt | $13 per cwt | 13½ cts. per lb | 4 cts. per lb. |
| Evaporated cream | $4.40 case of 4 doz. | $7.50 per case | $2.50 per doz | $20 per case | $13 per case | $12 per case | $4.50 to $5.50 per case. |
| Beans | 4 cts. per lb | 8 to 9 cts. per lb | $10 per cwt | 20 cts. per lb | 14 cts. per lb | $13.50 per cwt | 5 to 6 cts. per lb. |
| Ham | 16 cts. per lb | 25 cts. per lb | $40 per cwt | 60 cts. per lb | 40 cts. per lb | 30 cts. per lb | 18 cts. per lb. |
| Bacon | 18 to 20 cts. per lb. | 25 cts. per lb | $35 per cwt | 60 cts. per lb | 40 cts. per lb | 30 cts. per lb | 18 cts. per lb. |
| Fresh beef, by side | 11 cts. per lb. and up. | 25 cts. per lb | 30 to 50 cts. per lb. | | | | 25 to 35 cts. per lb. |
| Coffee | 15 to 35 cts. per lb. | 40 to 65 cts. per lb | 50 cts. per lb | 75 cts. per lb | 60 cts. per lb | 75 cts. per lb | 15 to 50 cts. per lb. |
| Tea | 25 to 60 cts | 40 to 65 cts. per lb | 75 cts. to $1 per lb. | $1.25 per lb | $1.05 per lb | 75 cts. to $1 per lb | 25 cts. to $1. per lb. |
| Salt | 2½ cts | 6 to 7½ cts. per lb | $10 per cwt | | | 10 cts. per lb | 4 cts. per lb. |
| Pickled pork | 12 cts. per lb. and up. | $25 per 100 lbs | | | | 30 cts. per lb | 13 to 15½ cts. per lb. |
| Potatoes, fresh | $1.75 per 100 lbs | 7 to 9 cts. per lb | $11 per cwt | | | 10 cts. per lb | 4 to 5 cts. per lb. |
| Onions, fresh | 2½ cts. per lb | 9 to 10 cts. per lb | $15 per cwt | | | | 4 to 5 cts. per lb. |
| Eggs, fresh | 30 cts. per doz | $12 to $15 per case | $18 per case of 30 doz. | | | $25 per case | $10 to $15 per case. |

[a] 5-inch pump, weighs 1,131 pounds; 6-inch pump, 1,417 pounds; 8-inch pump, 1,990 pounds; 10-inch pump, 3,130 pounds.
[b] Made with self-lubricating sheave, extra hammer, $7; extra sheave, $5.

TABLE 30.—*Prices of mining supplies in the North*—Continued.

| | Juneau. | Dawson. | Circle. | Mastodon post-office, Birch Creek region. | | Fairbanks. | Nome. |
|---|---|---|---|---|---|---|---|
| | | | | Summer. | Winter. | | |
| Hay | $35 to $40 per ton | $90 per ton | $100 per ton | | | $160 per ton | $50 per ton. |
| Oats | $40 per ton | $100 per ton | $110 per ton | | | $160 per ton | $50 per ton. |
| Overalls | 75 cts. per pair | $1.25 | $15 per doz | | | $1.80 | $1. |
| Jumpers | $1 each | $1.50 | $15 per doz | | | $1.80 | $1. |
| Leather miners' shoes | $3.75 per pair | $4 to $6 per pair | $4 to $7 per pair | | | $6 per pair | $4 to $12. |
| Gold Seal rubber boots | $8 per pair | $11 per pair | $10 | | | $12 | $8. |
| Gold Seal rubber shoes | $3 per pair | $3.50 per pair | $4.50 | | | $5 | $3.50 to $5. |
| Blankets (all wool) | $2.50 to $10 per pair | $10 | $5 to $15 | | | $10 | $8 to $15. |
| Duck | 12½ to 20 cts. per yd | 25 cts. to $1.25 per yd | 33⅓ to 60 cts. per yd | | | 50 cts. per yd | |
| Sweaters | $1.50 to $3.50 each | $2.50 to $6 | | | | $4.50 each | $2.50. |
| German socks | $0.75 to $1.50 per pair. | $1 per pair | $1 to $2 | | | $2.25 | $1.25 to $1.50. |
| Slicker coats | $2.50 each | $3.50 | | | | | |

## LEGAL CONDITIONS IN 1904.[a]

### INTRODUCTION.

After an investigation of the present conditions in the new so-called Fairbanks district of Alaska, situated on Tanana River, I was forcibly impressed with the urgent need of a change in the present location law, which allows the location of placer tracts by representation. An individual holding twenty or fifty powers of attorney from other citizens of the United States may go alone into a wild and unknown district and, by staking claims, each of 20 acres, tie up whole creek valleys which are by this process withdrawn from the area of public lands for a period generally amounting to eighteen months.

The location of the claims is then recorded in the nearest recording office in the names of the men who may be in San Francisco, Chicago, or New York. The ownership of the claims may afterwards be changed ad libitum by the passing of quitclaim deeds. The locator can have, under the law, but one claim by right of location on each creek; but by exercising the right of representation he withdraws from the public domain as many miles of creek beds as he can traverse and stake. On each claim he is required to do no work, beyond sinking a small discovery shaft, for the remainder of the year in which the location is made and up to the end of the ensuing year. Thus in any placer district in which the auriferous gravels have an economic value, and into which prospectors are continually pouring, as in the new Fairbanks district, many bona fide miners willing to search for gold to the extent of their physical and financial resources are rendered powerless through circumstances which they can in no way control or by any possibility foresee.

These men, willing to undergo the discomforts of an inhospitable country, either find themselves idle in a remote community, where living is expensive, or are compelled to make journeys of four or five days' duration from the base of supplies, and pack their provisions, blankets, and tools on their backs to find ground that has not been staked.

It may be objected that each man has an equal opportunity to reach a given piece of ground first. This, however, is hardly the case. The resources of the men who unite to send an individual to locate for them by power of attorney are generally greater than those of the individual who singly goes out with the purpose of locating one claim for his own operations. The power-of-attorney man, equipped with a better outfit and very likely dogs or horses, can go farther and stay longer than the prospector of limited resources.

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[a] I have discussed at some length the legal conditions of mining and the abuses prevailing under the present law in Alaska in a report made to the "Public Lands Commission," recently appointed by the President. In the same report I have embodied suggestions for a revision of the present law as far as it pertains to placer mines in Alaska.

Furthermore, the man who locates for others, if he does not represent persons living altogether at a distance, is frequently sent out by men whose interests are entirely inimical to the proper development of the region—namely, saloon keepers, nonproducers, "tin horns" (gamblers), or other riffraff such as are generally found in the new American mining camp. They have rarely the intention of operating the claims located in their names and thus producing gold. They hold the claims in the hope that the camp will boom, that inexperienced outsiders will come in, and that in the ensuing excitement they may sell at a figure based on the supposed value of the unprospected ground.

In the Fairbanks district portions of 3-mile stretches on three creeks are producing gold. Other portions within this producing area, as well as lengths of from 3 to 10 miles on six other creeks, are hardly prospected, are not producing, and are entirely withdrawn at present from exploration or exploitation through the process of the empowered staking above referred to. It is estimated that 2,000 persons will spend the winter in the district. Of these at least 1,200 will be able-bodied workingmen. The operations will afford work for possibly 600 men. One-half the remainder at least would prospect and develop new territory were it not for the nearly insuperable difficulties. The provisions of the present location law are, in my opinion, largely responsible for these difficulties.

In the portion of the Alaska interior lying between Yukon and Tanana rivers the auriferous gravels occupy a wide area. The overcoming of the purely physical obstacles to exploration is arduous, and many miles of creek valleys will be located as means of communication become established. If this large area and other areas yet to be explored are allowed to be progressively withdrawn from possible gold production, the result will be a distinct detriment to the development of Alaska.

A confirmation of the annoying and injurious effects of the law is afforded by the early history of the Nome gold district of Seward Peninsula, where the conditions were as follows:

> Nearly every individual located not only for himself but also for his many friends by power of attorney. To such an extent was the power of attorney here abused that more than 7,000 acres of ground were located, so that the several thousand Americans who arrived later, finding no unstaked ground anywhere in the vicinity, justly raised a somewhat bitter complaint.[a]

The practice of exercising the power of attorney is increasing in Alaska at present, and its abuse warrants the summary suspension of it if this be possible.

In September I visited the principal creeks on which gold-mining operations are being prosecuted in the Nome, Solomon, and Council districts of Seward Peninsula. Although the investigation of the legal conditions pertaining to gold mining was not called for by my instructions, certain facts regarding these conditions were impressed upon me. Indeed, it is impossible for an observer, impartially disposed, to go about the mining districts of Seward Peninsula and not seriously question the applicability of the present location law to Alaska conditions.

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[a] Schrader, F. C., and Brooks, A. H., Preliminary Report on the Cape Nome Gold Region, U. S. Geol. Survey, 1900, p. 32.

It is true that $100 worth of work is by law actually required on each 20 acres, but the statute is so loosely framed that the $100 worth of work is now done universally on each 160 acres, not only in Alaska but in all parts of the United States where placer mining is conducted. Moreover I have observed that this $100 worth of assessment work in all mining territory is little more than a farce. The suggestion that $100 on each 20-acre tract be used in making survey and plat of the claim the first year is worthy of consideration.

### APPLICATION OF THE PRESENT MINING LAW TO ALASKA.

The following order of the Secretary of the Interior, dated July 28, 1885 (Land Office Decisions, vol. 4, p. 128) extended the regulations of the General Land Office to Alaska.

> In pursuance of the eighth section of the act of Congress, approved May 17, 1884, entitled "An act to provide a civil government for Alaska" (23 Stat., 24), it is hereby prescribed that the rules and regulations of the General Land Office and Department of the Interior governing the administration of the mining laws of the United States be adopted for and extended to the district of Alaska, so far as the same may be applicable.

It is evident that the various provisions in Revised Statutes, Title XXXII, chapter 6, defining the United States mining laws, apply fully to Alaska.

> SEC. 2329. Claims usually called placers, including all forms of deposit, except veins of quartz, or other rock in place, shall be subject to entry and patent under like circumstances and conditions as are provided for vein or lode claims.

Lindley on Mines says (vol. 1, sec. 432) that this has been construed to mean:

> (1) That there must be a discovery upon which to base the location.
>
> (2) The location must be marked upon the ground so that its boundaries can be readily traced.

#### DISCOVERY.

According to the accepted interpretation of the law in the mining States of the United States, gold must be found on the ground in a discovery shaft before the claim can be validly held by location.

That many of the Alaska locations are illegal may be judged from the fact that very often three men have staked from ten to twenty 160-acre tracts of placer ground within periods varying from twenty-four to seventy-two hours. On every one of these tracts under the correct interpretation of the law, a shaft or drill hole at least 15 feet in depth would have been necessary in order to find one color of gold. Yet the same men have immediately recorded their staked ground as locations, swearing that they discovered gold. Lindley states (sec. 437): "The Land Department has uniformly held that discovery is essential in the case of placers, going so far at one time as to hold that such discovery was essential in each 20-acre tract within a location of 160 acres located by an association of persons." He further says: "Discovery is just as essential in case of placers as it is in lode locations."

The regulations of the General Land Office include the following:

> No lode claim shall be located until after the discovery of a vein or lode (or in a placer of gold) within the limits of the claim, the object of which provision is evidently to prevent the appropriation of presumed mineral ground for speculative purposes to the exclusion of bona fide prospectors before sufficient work has been done to determine whether a vein or lode (of gold) really exists.[a]

The nonobservance of the above requirement, and, in fact, the impossibility under the present law of its enforcement is the cause of the very evil in Alaska which the law aims to avert. I regard the promiscuous location and illegal holding of land on which gold has not been discovered as one of the most serious detriments to Alaskan development.

### SUBSEQUENT ANNUAL LABOR.

Regarding subsequent labor Lindley (sec. 624) makes the following statement:

> While a timely resumption of work may prevent a relocation, the law contemplates that the labor or improvements, actual and valuable, to the amount of $100 in each year, computed from the 1st day of January next succeeding the date of location, should be performed in good faith. There is probably no single provision of the law which is evaded to a greater extent than this one. While, of course, there are many exceptions the average locator exhausts his ingenuity in attempting to avoid this plain and wholesome requirement. The courts are disposed to deal with these drones in the hive with much more leniency than they deserve. The statute is too frequently applied on sentimental lines. Forfeitures, say these tribunals, are odious, and in many cases the reluctance with which they enforce the law encourages rather than deters the systematic evasion of it.
>
> The statute is extremely liberal as to the time in which the specified amount of work shall be performed. A location made on January 1, 1897, may, in the absence of State laws or local rules requiring development work to be performed as an act of location,[b] be held without a stroke of labor until December 31, 1898, and in no case

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[a] Mining laws and regulations, General Land Office, 1903, p. 27, §8.

[b] It should be borne in mind that the principal Western States where mining is conducted have, by State legislation, so enlarged upon the rather bare fabric of the Federal mining law that, within the boundaries of these States many of the abuses which are permissible under a liberal interpretation of the Federal law, have been done away with. On the other hand, the excellent restrictions in force in the placer-mining States have no force in Alaska, and in consequence decisions rendered by the courts must be much more liberal, and are productive of the evils for the obviation of which the State statutes were long ago enacted.—C. W. P.

is the period less than a full year. It would seem that a more rigid enforcement of the rule would not only command more respect for the law, but would in a great degree tend to promote the general object and intent of the mining statutes—the development of the mining resources of the country.

The above quotations are made because the opinions so closely bear on the deplorable conditions in Alaska. The climatic conditions are so formidable on the 31st of December of each year (the date of expiration of the time in which a miner may do his annual assessment work) that $100 might be expended by him in work on the claim in merely shoveling snow. Naturally, all traces of the work so done would be obliterated in a short time; yet it would be actual and valuable work, for otherwise he could not reach the ground beneath. This feature is so well understood in Alaska that a man who makes affidavit to work done holds his claim from year to year unquestioned, and it is undoubtedly true that the bulk of the territory held as placer ground is not only unworked but unvisited by the claimants after first location. Should, however, an individual relocate or "jump" a given claim so held, the first holder goes into court and swears that he has done his annual representation work, thus maintaining his title. Should the relocator maintain that he has searched the ground and found no traces of any previous work, he still has a weak case, and the verdict is usually against him. It is needless to say that this relocator has, in the great majority of cases, the better right to the ground.

I have endeavored to make clear a few of the reasons why the production of gold is actually retarded in Alaska, rather than assisted, by the present administration of the mining law.

## BOUNDARIES.

Of over a thousand prospectors in California, Oregon, Nevada, Colorado, Idaho, Montana, and Alaska whom I have known, talked with, and been in the field with, I have never seen ten who included a compass in their equipment, to say nothing of surveying instruments more nearly approaching precision. These men, generally known among themselves as "practical miners," are of the class by which mineral locations are generally made. Is it fair to assume that such men are competent to make a location which shall conform to the mining regulation?

His location notice should give the course and distance as nearly as practicable from the discovery shaft on the claim to some permanent well-known points or objects, such, for instance, as stone monuments, blazed trees, the confluence of streams, point of intersection of well-known gulches, ravines, or roads, prominent buttes, hills, etc., which may be in the immediate vicinity, and which will serve to perpetuate and fix the locus of the claim and render it susceptible of identification from the description thereof given in the record of locations in the district, and should be duly recorded.[a]

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[a] U. S. Mining Laws and Regulations—General Land Office, 1903, p. 27, sec. 9.

In Alaska, at least, the provisions called for by the above have become practically a dead letter, as is instanced by the case of the prospectors who during the season of 1903 were allowed to locate a group of claims defined as being on "Midas Creek, a tributary of a tributary of Koyukuk River." Midas Creek and the supposed locations afterwards proved to be entirely mythical.

The location law requires that the "location must be distinctly marked on the ground, so that its boundaries can be readily traced." How can the following decision of the Supreme Court of the United States, in the case of the McKinley Creek Mining Co. *v.* Alaska United Mining Co.,[a] which sustained the validity of a placer location where no attempt was made to actually mark the boundaries, be reconciled with the above? Lindley says:

> All that was done was to post notices on a snag or stump in a creek, claiming a certain number of feet running with the creek, and 300 feet on each side of the center of the creek, and referring to the claim as "the east extension of a certain-named claim and the west extension of another." Unless some facts or circumstances were represented to the court which can not be gleaned from the official report of the case, such a location would seem to fall short of the requirement that the claim shall be "distinctly marked on the ground, so that its boundaries may be readily traced."

I have no hesitation in saying that in Alaska, especially in Seward Peninsula, no attempt is ordinarily made by the locator to mark the boundaries of claims beyond the setting of stakes at the four corners averaging 3 feet in length by 1½ inches in diameter. These are generally overturned by the wind or uprooted by the frost within six months of the time of setting. And yet the burden of proof in case of subsequent dispute always rests on the relocator or the jumper.

I will call attention to the fact that the suggestions leading to modification in the present placer-mining law are directed to the reforming of the principal abuses which have been cited. Had time permitted, instances of abuse and misinterpretation of the present laws might have been indefinitely multiplied. Enough has been said to show that changes of a radical nature are needed in the laws and their administration, in order that the undoubtedly valuable mineral resources of Alaska may be exploited to the best advantage.

Much fault has been found in past years with the Territorial administration in the Klondike. That conditions affecting title have at times left much to be desired is undoubtedly true. At present efforts are being made to remedy the abuses which exist. In Alaska, on the other hand, each rich discovery of gold brings forth one or more lawsuits, and each new camp that is developed affords a profitable field of exploitation for swarms of parasites who, too indolent or too ignorant to devote themselves to bona fide mining, ingeniously strive, by manifold ways of trickery, to live by the labor of others.

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[a] Cited by Lindley, sec. 373.

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